



FLORIS OSMOND.

See Biographical Notice, p. 272.

TRANSACTIONS
OF THE
AMERICAN INSTITUTE OF MINING
ENGINEERS.

VOL. XLV.

CONTAINING THE PAPERS AND DISCUSSIONS OF THE
NEW YORK MEETING, FEBRUARY, 1913.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.

1914.

PREFACE.

THE technical papers and discussions contributed to the Institute during the year 1913 will be published in three volumes, of which this volume, containing the papers presented at the New York meeting of February, 1913, is the first.

The second of these volumes, to be entitled The Montana Volume, will contain the papers presented at the Butte meeting, August, 1913, describing the geology of the Butte district and the prevailing mining and metallurgical practice in Montana.

The third volume of the year will contain the remaining papers of the Butte meeting, as well as the papers presented at the New York meeting of October, 1913.

The Constitution and By-Laws, report of the Board of Directors, statistics of membership, and other official announcements which are published annually for the information of the members, and heretofore included in the *Transactions*, will be printed in the Year Book of the Institute.

CONTENTS.

Officers,	vii
Honorary Members,	viii
Committees,	ix
Institute Activities for the year 1913,	xiii

PROCEEDINGS.

New York Meeting, February, 1913,	xv
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PAPERS.

	PAGE.
The Ground-Waters. By JAMES F. KEMP (with Discussion),	3
The Sulphide Ores of Copper. Some Results of Microscopic Study. By L. C. GRATON and JOSEPH MURDOCH (with Discussion; see also p. 529),	26
The India Mica Industry. By A. FAISON DIXON,	94
The Elk City Mining District, Idaho County, Idaho. By ARTHUR L. FLAGG,	113
Fire-Clay Deposits of Canada. By HEINRICH RIES,	123
The Geographical Distribution of Mining Development in the United States. By EDWARD W. PARKER (with Discussion),	137
A Problem in Mining, together with Some Data on Tunnel-Driving. By F. M. SIMONDS and E. Z. BURNS (with Discussion),	147
Electric Power Installation at El Tigre, Sonora, Mexico. By JAMES W. MALCOLMSON,	189
The Hardinge Conical Mill. By H. W. HARDINGE,	194
Notes on the Formation of Ferrites in Roasting Blende. By G. S. BROOKS,	210
The Rôle of Certain Metallic Minerals in Precipitating Silver and Gold. By CHASE PALMER and EDSON S. BASTIN,	224
The London Mine, Mosquito Mining-District, Park County, Colo. By CHARLES J. MOORE,	239
School Laboratory-Work: Sampling of an Ore Containing Coarse Gold. By CHARLES E. LOCKE,	251
The Gay-Lussac Method of Silver Determination. By FREDERIC P. DEWEX,	256
Biographical Notice of Floris Osmond. By ALBERT SAUVEUR,	272
Valuation of Iron-Mines. By JAMES R. FINLAY (with Discussion),	282
Blast-Furnace Slag-Analyses for 24 Hours. By F. L. GRAMMER,	327
The Microstructure of Sintered Iron-Bearing Materials. By B. G. KLUGH,	330
New Design of Open-Hearth Steel-Furnace Using Producer-Gas. By HERBERT F. MILLER, JR. (with Discussion),	346
Notes on Cast-Iron. By ALBERT SAUVEUR (with Discussion),	355
The Production of Solid Steel Ingots. By BENJAMIN TALBOT (with Discussion),	400
Piping and Segregation of Ingots of Steel and Ductility-Tests for Open-Hearth Steel Rails. By P. H. DUDLEY,	424
The Use of Anti-Piping Thermit in Casting Steel Ingots. By E. A. BECK,	453
Commercial Production of Sound Steel Ingots. By EMIL GATHMANN,	461
Plant for Hadfield Method of Producing Sound Steel Ingots. By SIR ROBERT A. HADFIELD (with Discussion),	473
Sound Steel Ingots. (A Discussion),	481
Why Does Lag Increase with the Temperature from which Cooling Starts? By HENRY M. HOWE,	516
Comparative Notes on Steel-Rail Rolling. By ROBERT W. HUNT,	523

DISCUSSION.

Discussion of Messrs. Graton and Murdoch's Paper on The Sulphide Ores of Copper. Some Results of Microscopic Study. (See p. 26),	529
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Meetings of the Institute.

Three meetings of the Institute for the presentation and discussion of technical papers were held in 1913: The 104th meeting, comprising the Annual Business Meeting as well as technical sessions, was held at New York, Feb. 18 to 20, 1913; the 105th meeting was held at Butte, Mont., Aug. 16 to 21, 1913, at which papers obtained largely through the efforts of the Committee on Precious and Base Metals were presented; and the 106th meeting was held at New York, Oct. 16 and 17, 1913, under the auspices of the Iron and Steel Committee.

Local Sections.

In 1913, five local sections were established, as follows: St. Louis, Southern California, Colorado, San Francisco, and Montana. The Institute now has nine local sections.

Committees.

Seven technical committees were formed in 1913, on the following topics:

Use of Electricity in Mines,
Mining Geology,
Mining Methods,
Mining Law,
Petroleum and Gas,
Non-Metallic Minerals,
Coal and Coke.

This makes a total of nine technical committees.

A committee on Increase of Membership and a committee on Junior Members and Affiliated Student Societies were also formed in 1913.

Affiliated Student Societies.

The Mining Society of Pennsylvania State College, State College, Pa., an undergraduate organization, was recognized as an Affiliated Student Society in 1913. The total number of societies thus enrolled is 24.

Publications.

Transactions.—Volumes XLIII and XLIV, containing the papers and discussions at the meetings held in 1912, were distributed in 1913.

The Emmons Volume was also published in 1913.

Bulletin.—The *Bulletin*, which was issued regularly throughout the year 1913, contained 2,816 pages of technical papers and discussions, as compared with 1,588 pages in 1912; 436 pages of announcements, etc., in the preliminary section, and 294 pages of advertising matter; making a total of 3,546 pages of printed matter as compared with 2,253 pages in 1912.

Membership.

The total membership of the Institute on Jan. 1, 1914, was 4,150, as compared with 4,265 on Jan. 1, 1913. During 1913, 426 were added to the membership, as follows: 1 honorary member, 355 members, 31 associates, and 39 junior members. Against this should be recorded the deaths of 49 members and 1 associate; the resignation of 68 members and 9 associates; and the dropping from the rolls, for non-payment of dues, of 403 members and 11 associates—a total of 541. The unusual number dropped was due to the constitutional provision requiring all members to be dropped who are one year in arrears of dues.

Co-operation with Other Organizations.

The Institute was represented by delegates at the following conventions:

Canadian Mining Institute, March 5, 6, 7, 1913.

Twelfth International Geological Congress, Canada, Aug. 7–14, 1913.

American Mining Congress, Philadelphia, Pa., Oct. 20–24, 1913.

Fifth National Conservation Congress, Washington, Nov. 18–20, 1913.

Safety and Sanitation Congress, New York, N. Y., Dec. 10, 11, 12, 1913.

The International Engineering Congress of 1915 is to be held in connection with the Panama-Pacific Exposition, under the joint auspices of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, the Society of Naval Architects and Marine Engineers, and the American Institute of Mining Engineers.

Proceedings of the One Hundred and Fourth Meeting,
New York, N. Y., February, 1913.

COMMITTEES.

Committee at Large.—Executive Committee of the New York Section.

George F. Kunz, *Chairman.*

E. Gybbon Spilsbury, *Vice-Chairman.*

Thomas Robins.

H. J. Seaman.

Louis D. Huntoon, *Sec'y-Treas.*

Ladies.—Mrs. Thomas Robins, Chairman.

Mrs. George D. Barron,	Mrs. Louis D. Huntoon,	Mrs. Herman A. Prosser,
Mrs. Arthur S. Dwight,	Mrs. W. R. Ingalls,	Mrs. Charles F. Rand,
Mrs. Theodore Dwight,	Mrs. James F. Kemp,	Mrs. Frank McM. Stanton,
Mrs. Karl Eilers,	Mrs. Charles Kirchhoff,	Mrs. Bradley Stoughton,
Mrs. Arthur H. Elliott,	Miss Elizabeth H. Kunz,	Mrs. Otto Sussman.

Reception, Monday, Feb. 17, 1913.—George F. Kunz, Chairman.

W. deL. Benedict,	William W. Mein,	John R. Stanton,
William Campbell,	Ambrose Monell,	Edward B. Sturgis,
Theodore Dwight,	Herman A. Prosser,	Nat. Tyler, Jr.,
Arthur H. Elliott,	Thomas Robins,	Joseph A. Van Mater,
Louis V. Emanuel,	William L. Saunders,	Arthur L. Walker,
Lewis W. Francis,	E. Maltby Shipp,	William Y. Westervelt.

Reception, Tuesday, Feb. 18, 1913.—Arthur S. Dwight, Chairman.

Lawrence Addicks,	J. S. Lane,	Otto Sussman,
George B. Agnew,	H. S. Munroe,	Benjamin B. Thayer,
A. Chester Beatty,	A. J. Moses,	Kirby Thomas,
Charles A. Bohn,	W. H. Nicholls, Jr.,	Herbert G. Torrey,
W. R. Ingalls,	Charles Of,	A. H. Wethey,
John H. Janeway, Jr.,	E. A. C. Smith,	Edward L. Young.

Reception, Wednesday, Feb. 19, 1913.—Charles Kirchhoff, Chairman.

William Campbell,	Wheaton B. Kunhardt,	F. W. C. Schniewind,
Lewis W. Francis,	Richard Moldenke,	John M. Sherrerd,
Henry D. Hibbard,	C. Q. Payne,	Bradley Stoughton,
William Kent,	Joseph W. Richards,	Felix Vogel.

Banquet.—James Gayley, Chairman.

Edward B. Sturgis, *Vice-Chairman.*

George D. Barron,

Denis M. Riordan,

John D. Ryan,

W. deL. Benedict,

Thomas Robins,

William L. Saunders.

Luncheons.—E. Maltby Shipp, Chairman.

George M. Colvocoresses,

Frederick J. Pope,

Louis V. Emanuel,

Edward B. Sturgis.

The 104th meeting of the Institute was held at the Institute headquarters in the Engineering Societies Building, New York, N. Y., on Feb. 17, 18, and 19, 1913. During the sessions a registration bureau, bureau of information, and post-office were maintained under the charge of Mr. George Buckman.

Registration facilities were provided on Monday, Feb. 17. The opening session on that evening was in charge of the New York Section. Dr. George F. Kunz, *Chairman*, presided and delivered an address of welcome, which was followed by a very entertaining mock-serious account of the life of Mr. Charles F. Rand, President-elect, illustrated by lantern-slides, some of which were genuine and some palpably fictitious—the whole being done in Professor Kemp's famous style. At the conclusion of this address, the members and guests adjourned to the rooms of the Institute for further social intercourse and light refreshments. A pleasing feature of this occasion was the attendance of ladies, members of the Ladies' Committee and others.

The Annual Business Meeting of the Institute began at 10 a.m. on Tuesday morning, Feb. 18, 1913. The President announced, with deep regret, the death of John Fritz, the dean of the metallurgical profession, and of George W. Maynard, the dean of the mining profession.

At 12 o'clock noon, the retiring President, Prof. James F. Kemp, read his paper on *The Ground-Waters*.

At 2 p.m. the proceedings were continued by the presentation in oral abstract of the following papers:

**The Rôle of Certain Metallic Minerals in Precipitating Silver and Gold*, by Chase Palmer and E. S. Bastin, Washington, D. C. (Presented by E. S. Bastin.)

**A Problem in Mining, Together with some Data on Tunnel-Driving*, by F. M. Simonds and E. Z. Burns, New York, N. Y. (Presented by E. Z. Burns and discussed by William L. Saunders, James F. Kemp, Henry S. Munroe, and E. Z. Burns.)

**The Geographical Distribution of Mining Development in the United States*, by E. W. Parker, Washington, D. C. (Discussed by James F. Kemp.)

*Electric Power Installation at El Tigre, Sonora, Mexico, by J. W. Malcolmson, Kansas City, Mo.

Structure of the Northern Anthracite Basin Relative to Forms of Folds, by N. H. Darton.†

*The Sulphide Ores of Copper: Some Results of Microscopic Study, by L. C. Graton, New York, N. Y., and Joseph Murdoch, Cambridge, Mass. (Presented by L. C. Graton and discussed by James F. Kemp, Horace V. Winchell, Alfred J. Moses, Edson S. Bastin, and L. C. Graton.)

A discussion of Prof. James F. Kemp's paper on The Ground-Waters, by N. H. Darton, C. Q. Payne, and James F. Kemp.

The following papers were read by title for future publication by the Institute:

*Fire-Clay Deposits of Canada, by Heinrich Ries, Ithaca, N. Y.

*The Hardinge Conical Mill, by H. W. Hardinge, New York, N. Y.

Notes on the Formation of Ferrites in Roasting Blende, by G. S. Brooks, Depue, Ill.

The Gay-Lussac Method of Silver Determination, by F. P. Dewey, Washington, D. C.

The India Mica Industry, by A. F. Dixon, New York, N. Y.

*Elk City Mining-District, Idaho County, Idaho, by A. L. Flagg, Spokane, Wash.

*School Laboratory-Work; Sampling of an Ore Containing Coarse Gold, by Charles E. Locke, Boston, Mass.

*The London Mine, Mosquito Mining-District, Park County, Colo., by C. J. Moore, Denver, Colo.

The New York Section Committee provided an illustrated lecture on Tuesday evening, Feb. 18, by Frederick Haynes Newell, Director of the United States Reclamation Bureau, on Irrigation and Its Relation to the Mining Industry. Assembly Room No. 1, on the fifth floor, was well filled, and the audience were repaid for their attendance by an address of unusual interest, illustrated by colored lantern-slides of more than ordinary beauty. At the close of the lecture an informal collation

* Available for distribution in printed form.

† Not furnished for publication.

was served in the Library on the thirteenth floor, with the Librarian, Mr. W. P. Cutter, and his assistants, in attendance.

The second session, on Wednesday morning, Feb. 19, was devoted to Iron and Steel. Printed copies of all papers were distributed at 9 a.m., and Prof. James F. Kemp called the meeting to order at 10 a.m. There were many non-members of the Institute present as guests. The following papers were presented in oral abstract by the authors, except where otherwise noted:

*Piping and Segregation of Ingots of Steel and Ductility-Tests for Open-Hearth Steel-Rails, by P. H. Dudley, New York, N. Y.

*The Use of Anti-Piping Thermit in Casting Steel Ingots, by E. A. Beck, New York, N. Y.

*The Production of Solid Steel Ingots, by Benjamin Talbot, Middlesbrough, England. (Presented by C. Kirchhoff.)

*Commercial Production of Solid Steel Ingots, by Emil Gathmann, Baltimore, Md.

Plant for Hadfield Method of Producing Sound Ingots, by Sir Robert A. Hadfield. (Presented by Henry D. Hibbard.)

The general discussion was opened by Prof. Albert Sauveur, who was followed by Robert Job, representing the Lehigh Valley R. R.; Max W. Wickhorst, Engineer of Tests, American Railway Engineering Association; L. E. Howard, Metallurgist, Simonds Engineering Co.; J. E. Sague, of the Public Service Commission; James E. Howard, Engineer Physicist, U. S. Bureau of Standards; W. C. Cushing, Chief Engineer, Maintenance of Way, Pennsylvania R. R.; Guillian H. Clamer; Dr. Allerton S. Cushman, representing the American Rolling Mill Co.; N. Petinot, representing the Titanium Alloy Manufacturing Co.; Prof. Henry M. Howe; Karl W. Zimmerscheid; Frederick W. Wood; Dr. P. H. Dudley; Henry Hess.

The third technical session was held at 2 p.m. on Wednesday, Feb. 19, Past President John Birkinbine in the chair. The following papers were presented:

* Available for distribution in printed form.

*Valuation of Iron-Mines, by James R. Finlay, New York, N. Y. (presented in oral abstract by the Secretary of the Iron and Steel Committee and discussed by John Birkinbine, Harrison Souder, J. W. L. Birkinbine, B. W. Vallat, R. V. Norris, and E. W. Parker.

*Notes on Cast-Iron, by Albert Sauveur, Cambridge, Mass. (Discussed by Prof. H. M. Howe and Dr. Richard Moldenke.)

*Why Does Lag Increase with the Temperature from which Cooling Starts? by Henry M. Howe, New York, N. Y. (Discussed by Prof. Albert Sauveur.)

*The Microstructure of Sintered Iron-Bearing Materials, by B. G. Klugh, Birdsboro, Pa. (Presented by Arthur S. Dwight.)

*New Design of Open-Hearth Steel-Furnace Using Producer-Gas, by Herbert F. Miller, Jr., Verona, Pa. (Discussed by Henry D. Hibbard and Herbert F. Miller, Jr.)

The following papers were read by title for future publication by the Institute:

*Comparative Notes on Steel-Rail Rolling, by Robert W. Hunt, Chicago, Ill.

Biographical Notice of Floris Osmond, by Albert Sauveur, Cambridge, Mass.

*Blast-Furnace Slag-Analyses for Twenty-four Hours, by F. L. Grammer, Leesburg, Va.

Annual Banquet.

The Annual Banquet, preceded by a reception to the new and retiring Presidents of the Institute, was held at the Waldorf-Astoria on Wednesday evening, Feb. 19, 1913, and was attended by 146 members and guests. After the Banquet, Prof. James F. Kemp, in a charming address, which he called a swan song, introduced the new President, Charles F. Rand, who made an address of greeting, expressing the good-will and friendly co-operation which has characterized all his efforts for the Institute.

Professor Kemp then expressed the gratitude of the Institute to the Secretary of the New York Section, Prof. Louis D. Hun-

toon, through whose efforts the social and business part of the meeting had been rendered the success which it undoubtedly was; and to Mr. Edward B. Sturgis, who had so ably conducted the preliminaries leading to a most enjoyable banquet.

Professor Kemp then presented Mr. William L. Saunders, who presided as Toastmaster in his usual felicitous manner.

The Toastmaster first introduced Dr. A. E. Barlow, President of the Canadian Mining Institute, who was followed by Charles Warren Hunt, Secretary of the American Society of Civil Engineers, and H. M. Chance, President of the Mining and Metallurgical Society of America.

An unusual feature of the program was inaugurated when the Toastmaster introduced Miss Inez Mulholland, who held the attention of the audience very agreeably for some minutes on the subject of Equal Suffrage. She was followed by Mr. Denis M. Riordan, who sang a song of his own composition, which met with great applause; and who also read the following poem and a letter of regret sent by Dr. R. W. Raymond:

“ The Ladies.”

By a Regretful and Disappointed Absentee Post-Prandial Orator

February 19, 1913

How many times have I essayed
This ancient sentiment to honor,
Naming the matron or the maid,
And heaping compliments upon her!

I've quoted orators of old
Or tried to be their true successor,
Ending the ardent tale I told
With “Woman!” followed by
“God bless her!”

Alas! for all my garnered store—
The tunes of bards, the words of sages—
To this new age they fit no more,
No matter what the women's age is!

Something is weirdly changed, I trow,
Since I was wont, in places shady,
To urge my suit, and breathe my vow!
Am *I* thus changed, or is the Lady?

The One I knew was full of skill
 To soothe life's pains amid life's clamor ;
 This One just hollers louder still,
 While on the panes she plies a hammer !

The One I knew was glad to get
 Love's letters ; but the paradox is,
 That this One pours, with purpose set,
 Green paint into Love's letter-boxes !

The One I knew inspired men's souls
 And unto deeds heroic bound them ;
 This One goes hiking for the polls
 Just after men have up and found them !

Pray, with the Old Man sympathize
 Who needs must face the choice pathetic :
 Either take pepper in his eyes,
 Or make his language suffragettic !

Such nightmares do my spirit vex !
 "The Ladies !" How should I address 'em ?
 Yet, even as the Stronger Sex,
 I still, as always, say "God bless 'em !"

The Toastmaster then called upon Mr. P. N. Moore, and a brilliant evening was closed with an address by Dr. James Douglas.

*List of Members and Guests (Doubtless Incomplete) Registered
 at Institute Headquarters.*

Acker, E. O'C., Bethlehem, Pa.	Bates, F. A., New York, N. Y.
Adams, Walter C., Durango, Mexico.	Bates, Moury, New York, N. Y.
Addicks, Lawrence, Chrome, N. J.	Beale, A. H., Vandergrift, Pa.
Aertsen, G., Philadelphia, Pa.	Beck, E. A., New York, N. Y.
Ahles, R. L., Williamsport, Pa.	Bellinger, H. P., Syracuse, N. Y.
Anderson, R W., Pittsburg, Pa.	Benedict, W. de L., New York, N. Y.
Ashley, E. B., New York, N. Y.	Berggreen, Paul H., New York, N. Y.
Ayres, W. S., Hazleton, Pa.	Bohn, C. A., New York, N. Y.
Ayres, Mrs. W. S., Hazleton, Pa.	Birkinbine, J. L. W., Philadelphia, Pa.
Backert, O. A., Cleveland, Ohio.	Birkinbine, John, Philadelphia, Pa.
Banks, N. C., Detroit, Mich.	Boylston, H. M., Ridgewood, N. J.
Barba, W. P., Midvale, Pa.	Boynton, Henry C., Trenton, N. J.
Barbour, Percy E., Candor, N. C.	Brock, R. W., Ottawa, Canada.
Barlow, Alfred E., Montreal, Canada.	Brookfield, W. B., Syracuse, N. Y.
Barren, H. A., Cleveland, Ohio.	Brown, Thomas E., New York, N. Y.
Barren, Harry B., Cleveland, Ohio.	Browne, David H., Copper Cliff, Ont.
Barron, George D., Rye, N. Y.	Browne, de Courcy, New York, N. Y.
Barron, Mrs. Mabel W., Rye, N. Y.	Burden, J. A., Troy, N. Y.
Bastin, E. S., Washington, D. C.	Burdick, C. A., New York, N. Y.

- Burns, E. Z., New York, N. Y.
 Campbell, William, New York, N. Y.
 Carey, Charles D., So. Bethlehem, Pa.
 Carney, F. D., Steelton, Pa.
 Chance, H. M., Philadelphia, Pa.
 Church, John A., New York, N. Y.
 Clamer, G. H., Philadelphia, Pa.
 Clymer, E. T., Philadelphia, Pa.
 Colvocoresses, G. M., New York, N. Y.
 Corning, C. R., New York, N. Y.
 Corning, Edwin, Albany Co., N. Y.
 Cox, Herbert B., Calais, Me.
 Coyle, John A., Bridgeton, N. J.
 Cushing, W. C., Pittsburg, Pa.
 Cushman, A. S., Middletown, Ohio.
 Darton, N. H., Washington, D. C.
 Darton, Mrs. N. H., Washington, D. C.
 Davidson, G. M., Chicago, Ill.
 Day, Mrs. H. N.
 de Hora, M. H., New York, N. Y.
 De Maré, Baltzar E. L., Philadelphia, Pa.
 Devereux, J. H., New York, N. Y.
 Dieffenbach, E. E., Newark, N. J.
 Dorr, John V. N., Denver, Colo.
 Douglas, James, New York, N. Y.
 Du Bon, Thomas W., Philadelphia, Pa.
 Dudley, P. H., New York, N. Y.
 Dufourcq, Edward L., New York, N. Y.
 Dutrick, J. W., Youngstown, Ohio.
 Dwight, A. S., New York, N. Y.
 Dwight, Mrs. A. S., New York, N. Y.
 Dwight, Theodore, New York, N. Y.
 Dwight, Mrs. Theodore, New York, N. Y.
 Eilers, A., Brooklyn, N. Y.
 Eilers, Mrs. A., Brooklyn, N. Y.
 Eilers, Miss E., Brooklyn, N. Y.
 Elliott, Arthur H., New York, N. Y.
 Elliott, Mrs. A. H., New York, N. Y.
 Emanuel, L. V., Perth Amboy, N. J.
 Emlaw, H. S., Brooklyn, N. Y.
 Fallon, B. J., Chicago, Ill.
 Faust, Edward S., New York, N. Y.
 Finch, John W., Denver, Colo.
 Finch, Mrs. John W., Denver, Colo.
 Frank, Dr. Karl Georg, New York, N. Y.
 Garlichs, H., Newark, N. J.
 Gatlin, T. H., Washington, D. C.
 Goodrich, R. R., Tucson, Ariz.
 Goodspeed, G. M., McKeesport, Pa.
 Gathmann, Emil, Baltimore, Md.
 Gennet, C. W., Jr., Chicago, Ill.
 Gouyard, G. M., New York, N. Y.
 Grammer, F. L., Leesburg, Va.
 Granger, A. O., Cartersville, Ga.
 Graton, L. C., Cambridge, Mass.
 Groome, A. C., Baltimore, Md.
 Gummere, W., Roebbing, N. J.
 Hall, John H., New York, N. Y.
 Hamilton, W. H., New York, N. Y.
 Hammond, John Hays, New York, N. Y.
 Hardinge, H. W., New York, N. Y.
 Hawk, W. B. N., Lorain, Ohio.
 Helander, A. H., Youngstown, Ohio.
 Henry, A. S., New York, N. Y.
 Hess, Henry, Philadelphia, Pa.
 Hibbard, H. D., Plainfield, N. J.
 Hofman, H. O., Boston, Mass.
 Holbrook, L., New York, N. Y.
 Howard, James E., Washington, D. C.
 Howard, L. E., Lockport, N. Y.
 Howe, Henry M., Bedford Hills, N. Y.
 Howells, George A., New York, N. Y.
 Humphrey, George S., New York, N. Y.
 Hunnings, S. V., Schenectady, N. Y.
 Hunt, C. W., New York, N. Y.
 Huntoon, L. D., New York, N. Y.
 Huntoon, Mrs. L. D., New York, N. Y.
 Ihlseng, A. O., New York, N. Y.
 Ihlseng, Mrs. A. O., New York, N. Y.
 Ingalls, W. R., New York, N. Y.
 Ingalls, Mrs. W. R., New York, N. Y.
 Iredell, Frank W., New York, N. Y.
 Jackson, S. W.
 Job, Robert, Montreal, Canada.
 Jobson, T., Harrison, N. J.
 Jones, W. L., Pittsburg, Pa.
 Jouet, C. H., New York, N. Y.
 Kemp, James F., New York, N. Y.
 Kemp, Mrs. James F., New York, N. Y.
 Kemp, Miss K., New York, N. Y.
 Kennedy, John S., Stanhope, N. J.
 Kenney, E. F., Johnstown, Pa.
 Kent, William, Montclair, N. J.
 Kidder, F. J., New York, N. Y.
 Kirchhoff, C., New York, N. Y.
 Kirchhoff, Mrs. C., New York, N. Y.
 Kreuzpointner, Paul, Altoona, Pa.
 Kroeger, Herbert B., Perth Amboy, N. J.
 Kunz, George F., New York, N. Y.
 Landemuth, L. B., Steelton, Pa.
 Lane, J. S., New York, N. Y.
 Lanza, G., Philadelphia, Pa.
 Ledoux, A. R., New York, N. Y.
 Ledoux, Mrs. A. R., New York, N. Y.

- Leonard, William A., Wareham, Mass.
 Liddell, Donald M., New York, N. Y.
 Lyon, D. A., Pittsburg, Pa.
 McCreery, J. Harold, Plainfield, N. J.
 McKune, F. B., Hamilton, Canada.
 McMahon, F. J., Wilkes-Barre, Pa.
 Macon, William W., New York, N. Y.
 Malcomson, James W., Kansas City, Mo.
 Marshall, C. S., Worcester, Mass.
 Martin, C. H., Cleveland, Ohio.
 Masters, Harris K., Salt Lake City, Utah.
 Mathews, John A., Syracuse, N. Y.
 Mead, F. H., Syracuse, N. Y.
 Mellen, G., E. Orange, N. J.
 Merriman, Mansfield, New York, N. Y.
 Michel, William, Columbus, Ohio.
 Miller, A. J., New York, N. Y.
 Miller, Herbert F., Jr., Verona, Pa.
 Mills, C. P., Portsmouth, Ohio.
 Moldenke, R., Watchung, N. J.
 Moore, L. D., Chatham, N. J.
 Moore, Philip N., St. Louis, Mo.
 Morrow, John G., Hamilton, Canada.
 Morse, Henry G., New York, N. Y.
 Moses, Alfred, Jr., New York, N. Y.
 Mosman, P. A., New York, N. Y.
 Munroe, Henry S., New York, N. Y.
 Murray, C. B., Cleveland, Ohio.
 Murray, Mrs. C. B., Cleveland, Ohio.
 Neustaedter, A., Jersey City, N. J.
 Newell, F. H., Washington, D. C.
 Nichols, W. H., Jr., New York, N. Y.
 Norris, R. V., Wilkes-Barre, Pa.
 Of, Charles, Palisade, N. J.
 Olcott, E. E., New York, N. Y.
 Orr, J. F., New York, N. Y.
 Osborne, C. G., So. Chicago, Ill.
 Overpeck, A. C., Hill City, S. D.
 Owens, R. B., Philadelphia, Pa.
 Paine, F. W., Boston, Mass.
 Parker, E. W., Washington, D. C.
 Patton, Albert, McKeesport, Pa.
 Patterson, R. C., Jr., New York, N. Y.
 Payne, C. Q., New York, N. Y.
 Pease, F. N., Altoona, Pa.
 Petinot, N., Niagara Falls, N. Y.
 Pierrefeu, A. L., Chicago, Ill.
 Pitman, S. M., Providence, R. I.
 Prosser, H. A., New York, N. Y.
 Prosser, Mrs. H. A., New York, N. Y.
 Ramage, J. C., Washington, D. C.
 Rand, C. F., New York, N. Y.
 Rand, Mrs. C. F., New York, N. Y.
 Rand, Miss, New York, N. Y.
 Ray, F. A., St. Louis, Mo.
 Raymer, G. S., Cambridge, Mass.
 Reinhardt, G. A., Newark, N. J.
 Rice, George S., Pittsburg, Pa.
 Rich, Charles H., Conshohocken, Pa.
 Richards, Joseph W., S. Bethlehem, Pa.
 Ricketts, L. D., Cananea, Mexico.
 Riordan, D. M., New York, N. Y.
 Riter, George W., New York, N. Y.
 Robinson, C. S., Youngstown, Ohio.
 Robinson, T. W., Chicago, Ill.
 Rodgers, S. M., Pittsburg, Pa.
 Roeber, E. F., New York, N. Y.
 Rolle, Sidney, Chrome, N. J.
 Rowe, W. H., Pittsburg, Pa.
 Rys, C. F. W., Pittsburg, Pa.
 Sague, J. E., Albany, N. Y.
 Sargent, George W., Pittsburg, Pa.
 Satterthwaite, George, Philadelphia, Pa.
 Saucton, A. W., Montreal, Canada.
 Saunders, W. L., New York, N. Y.
 Sauveur, Albert, Cambridge, Mass.
 Savage, J. R., Jamaica, N. Y.
 Schneider, Albert F., New York, N. Y.
 Schniewind, F., New York, N. Y.
 Seddon, William L., Norfolk, Va.
 Sharpless, F. F., New York, N. Y.
 Sherrerd, J. M., New York, N. Y.
 Shimer, W. R., Bethlehem, Pa.
 Shipp, E. Maltby, New York, N. Y.
 Slocum, A. N., Pittsburg, Pa.
 Simonds, A. T., Lockport, N. Y.
 Simonds, Daniel, Fitchburg, Mass.
 Simonds, F. M., New York, N. Y.
 Smith, E. Percy, New York, N. Y.
 Smith, J. William, Syracuse, N. Y.
 Souder, Harrison, Cornwall, Pa.
 Speller, Frank N., Pittsburg, Pa.
 Stanton, F. McM., New York, N. Y.
 Stark, Charles W., New York, N. Y.
 Stark, Mrs. W. N., Worcester, Mass.
 Stein, C. H., Jersey City, N. J.
 Stephenson, Henry, New York, N. Y.
 Stevenson, A. A., Philadelphia, Pa.
 Stone, George C., New York, N. Y.
 Stoughton, Bradley, New York, N. Y.
 Stoughton, Mrs. Bradley, New York, N. Y.
 Struthers, Joseph, New York, N. Y.
 Sturgis, Edward B., New York, N. Y.
 Sturgis, Mrs. Lyman B., New York, N. Y.

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| Sussman, Otto, New York, N. Y. | Wellman, S. T., Cleveland, O. |
| Sussman, Mrs. Otto, New York, N. Y. | Wentworth, Henry A., Boston, Mass. |
| Taylor, Knox, High Bridge, N. J. | Wethey, A. H., Cambridge, Mass. |
| Thorp, G. G., Chicago, Ill. | White, Charles H., Cambridge, Mass. |
| Torrey, Herbert G., New York, N. Y. | White, G. A., Pittsburg, Pa. |
| Torrey, Mrs. Herbert G., New York, N. Y. | Wickhorst, M. H., Chicago, Ill. |
| Treat, L. B., Glastonbury, Conn. | Wiley, William H., New York, N. Y. |
| Turner, N. P., Cuba. | Wilmer, E. G., Milwaukee, Wis. |
| Uehling, Ed. A., Passaic, N. J. | Williams, David, New York, N. Y. |
| Unger, J. S., Pittsburg, Pa. | Wilmot, F. A., Bridgeport, Conn. |
| Vallat, B. W., Ironwood, Mich. | Winchell, H. V., Minneapolis, Minn. |
| Vauclain, Parry, Philadelphia, Pa. | Withington, Howard P., New York, N. Y. |
| Vogel, Felix A., New York, N. Y. | Wood, F. W., Sparrows Point, Md. |
| Vollmer, William, Burnham, Pa. | Woodman, J. E., New York, N. Y. |
| Walker, Arthur L., New York, N. Y. | Yang, Cho, So, Bethlehem, Pa. |
| Walker, W. R., New York, N. Y. | York, Howard P., Brooklyn, N. Y. |
| Wang, Y. Tsenshan, New York, N. Y. | York, James E., Brooklyn, N. Y. |
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P A P E R S.

The Ground-Waters.*

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(New York Meeting, February, 1913.)

IN the study of ore deposits water plays a very prominent part. The small group of igneous ores involve it least, but in all the rest it is an invariable feature. The ground-water, therefore, is perhaps as appropriate a subject for discussion before a session of the American Institute of Mining Engineers as any topic which might be selected. The theme is if anything the more appropriate, because two very important contributions to it have already appeared in the *Transactions*. The great essay in 1893 of Franz Posepny,¹ systematized and classified the subject, and was doubtless not without its influence in leading to the presentation in 1900, by C. R. Van Hise, of an important contribution entitled *Some Principles Controlling the Deposition of Ores*.² We all recall the discussions of 10 or 12 years ago, and the interest in these themes which then prevailed in the Institute. They are collectively accessible in the separate edition wisely assembled by Dr. Raymond, and now to be supplemented by the Emmons Volume. Views have been progressing in the past decade or more, and new evidence has been brought to light. A general summary and concise statement may, therefore, not be without interest.

There are three kinds of waters which command attention: *The Meteoric*, which is derived primarily from the rainfall; *The Connate*,³ which is deposited in the interstices of marine or fresh-water sediments, and is buried with them; and *The Magmatic*, called also *Juvenile*, or the water derived from igneous magmas. The oceanic waters have generally been regarded as

* Address of the retiring President.

¹ The Genesis of Ore-Deposits, *Trans.*, xxiii., 197 to 369 (1893).

² *Trans.*, xxx., 27 to 177 (1900).

³ The name connate is less familiar than the other two. It was suggested by A. C. Lane in *Mine Waters and Their Field Assay*, *Bulletin of the Geological Society of America*, vol. 19, pp. 501 to 512 (1907). It has proved distinctly useful.

essentially of meteoric origin. The ceaseless contributions of the rivers, obviously supported by the rainfall, have given rise to this view. The dissolved salts of the ocean have been attributed to the solution of minerals on the land, and their introduction into the sea by the drainage. The age of the earth has even been computed by dividing the total of dissolved solids in the ocean by the estimated annual contributions brought in by the rivers. The ocean waters, however, as the venerable and revered Austrian geologist, Edouard Suess, once suggested, may have been derived, together with much of their dissolved salts, directly from ancient volcanoes. If so, the calculations of the age of the earth in the manner just mentioned have little significance.

Other views have also been held of oceanic waters, not entirely concerned with their first entrance into the substance of the globe. That the connate waters in the Palæozoic strata reproduce for us to-day the composition of the ancient Palæozoic sea, was a favorite theme of the late T. Sterry Hunt, a Past-President of the Institute. The connate waters are notably rich in calcium, as will be later remarked; but by contrast R. A. Daly has argued in favor of the limeless ocean of the pre-Cambrian times, despite the fact that what seems to be the greatest section of limestone in North America is in the Grenville series of the pre-Cambrian strata of Ontario. Apparently overlooking this fact, Dr. Daly argued that only as lime became abundant in the ancient seas were animals able to secrete hard parts, and leave fossil remains. Hence our failure to find fossils in the oldest rocks, and their sudden appearance in the Cambrian. Others have even suggested that the salty solution which forms the basis of the blood in all forms of animals, and which is singularly alike in them all, represents the composition of the remote oceanic medium in which the ancestors of modern forms of life first had their being. Originally entering into their circulations, it has ever since been maintained, because suited to the purpose.

We may, therefore, leave the oceanic waters with a query as to their nature.

1. *The Meteoric Waters.*—The rain falls upon the earth in different annual amounts in different parts of the world. In some extremely arid districts its quantity is very small; and at best

in such regions it is very irregular. We find sometimes almost continuous drought, broken at long intervals by brief, but violent cloudbursts. In less arid areas the contributions of the clouds to the earth are greater, and in certain portions of the Puget Sound region and of Florida they reach a maximum for this country. As is generally known, the maximum annual rainfall at any point in the United States is above 85 in.; the minimum is less than 5. The general range in the eastern half is from 20 to 40 inches.

When the rainfall reaches the earth it divides into three parts. A part returns directly into the atmosphere by evaporation, while yet the drops are passing through the air or are standing on leaves or other surfaces. A part flows away in small and large streams towards, and finally into the ocean, experiencing in transit additional evaporation. This portion is called the run-off. The proportional determination of it made by Humphreys and Abbott, years ago, for the Mississippi was 25 per cent. for the entire drainage basin. For individual streams it ranged from 20 to 90 per cent. of the local rainfall. We have innumerable additional and later measurements of these and other streams, but the figures will serve for a general illustration.

The third portion of the rainfall sinks into the ground, unless it falls upon bodies of standing water. This descending portion passes along various larger or smaller passages, until it joins that standing body underground which is called the ground-water. There is reason for the conclusion that where the colder climates prevail the melting snows at the breakup of winter are the most efficient contributors to the ground-water. At this season the surface-waters stand for the longest periods and in the greatest amount, before they run off, and have the best opportunity to percolate downward. Wells have highest water in the spring. The upper surface of this underground reservoir we name the ground-water level. It is the horizon at which wells first encounter water and at which miners in sinking shafts are forced to pump. In the regions of average rainfall the surface of the ground-water is a few feet down and it roughly conforms to the slope of the ground, but is flatter than hillsides. Near summits it is farther from the surface than in valleys. In swamps and lakes it is at the surface. It is also

conditioned in part by the structure and porosity of the rocks. The upper portion of the ground-water is fed out to the surface again in springs, which in the end join the waters of the runoff. A suitable orifice leading to the surface and a storage reservoir of ground-water above the vent are all that are needed to produce them. With irregular vertical distribution of the ground-water a siphonic course may be locally followed which may drop below the general ground-water level. Interesting instances are figured in Posepny's essay in the *Transactions* of 1893.

Between the ground-water level and the surface lies a zone which is traversed by the descending, oxidizing waters and which is characterized by extreme alteration of both rocks and contained minerals. This zone, though long known by miners to be the home of the oxidized ores, was acutely emphasized by Posepny and called the "vadose" region, a name which, because of usefulness and priority, we may maintain. To much the same zone Van Hise applied the name "belt of weathering,"⁴ and J. W. Finch called it the "gathering zone,"⁵ implying thereby that through it the ground-waters were fed. The ground-water level lies very deep in arid regions. The mines at Mapimi, Durango, Mexico, encountered no water as they followed down the lead-silver shoots in limestones until they were 2,400 ft. from the surface. The Horn-Silver mine, in southwestern Utah, reached a vertical depth of 600 ft. without encountering water except in one small spring on the third level.⁶ We do not know how much deeper the ground-water lies. At other places in arid regions different experiences are met. If, for instance, we have porous gravels in a pocket of impervious rocks, there may be large supplies of water, as in the so-called "bolsons" of the Southwest and Mexico. The metallurgical center of Douglas, Ariz., is located on one. The bolson determined the establishment of the smelter, since the first requisite for a reduction works is not ore or fuel, but water. Elsewhere in the arid district where mines have been sunk in limestone, pockets of water have been struck in caves. At Eureka, Nev., as we learn from the older engineers, when en-

⁴ *Trans.*, xxx., 72 (1900).

⁵ *Proceedings of the Colorado Scientific Society*, vol. vii., p. 201 (1902-04).

⁶ Private communication from B. B. Lawrence.

countered in upper workings, they were allowed to run away to lower levels through natural passages, such as are sometimes encountered in mines in limestone. Eventually, however, with increasing depth, a level was reached apparently with no lower cavities and the waters had to be pumped.⁷

Reference may also be made to the recent experience at Tombstone, Ariz., in which great quantities of water were encountered. The pumps yielded a heavy outflow, although the region is one of small rainfall. Yet the infrequent storms had sufficed to produce great quantities of stored-up water, although apparently at a long distance from the mines. There was no good reason to refer the water to any other source than to the rainfall.

We all recall the experience on the Comstock where at depths of 2,200 ft. in the Savage shaft, hot waters were tapped by breaking through tight walls of gouge. Despite heroic efforts the abundance of hot water and the high temperature finally made sinking impracticable below the extreme depth of 3,080 ft. as reached in the Yellow Jacket shaft. Observers who have written of the Comstock generally concluded that these waters had come from the rainfall which had percolated to depths whence the heat of previous eruptive action had not been dissipated. With greatly elevated temperatures they had become confined in the closed cavities where tapped. The waters were still under head, for when first encountered at 2,200 ft. in the Savage shaft they rose quickly 450 ft. The case of the Comstock is less simple than some others because of the presence of eruptive rocks with their attendant heat-phenomena. There is always the possibility that at least some of the water may have come directly from them. To this point attention will be again given.

We may say in summary that in regions of good rainfall the ground-water stands at slight or moderate depth. In regions of small rainfall it is irregular in distribution; it may be pockety; it may not be met until great depths. The most interesting feature of the ground-water is not its upper limit, but its lower boundary. To this we will later return after reviewing the other two varieties, the connate and the magmatic.

⁷ As learned from R. M. Catlin.

2. *The Connate Waters.*—Sands and finer sediments cannot well be deposited beneath the ocean and its embayments, or beneath fresh water, without involving in their interstices portions of the medium in which they are laid down. As they become buried under an increasing load this water may in part be squeezed out, especially from shales; it may in part enter into hydrated silicates formed in metamorphism, but in part it doubtless remains in the interstices. To this source we are accustomed to assign the brines entombed in ancient sandstones such as the Berea grit of the Lower Carboniferous strata of Ohio; the brines of other horizons in America and abroad, and some minor amounts of fresh water which are not so abundant. Should a lava-flow pour out on the bottom of a body of fresh water or of salt water, and be blown up into a porous condition by expanding steam or other gases, it would become saturated with the surrounding waters, and possibly carry them down with it in burial. The waters in the deeply-buried lava sheets of Keweenaw Point have been referred to this source. Special interest has attached to samples of connate waters obtained in deep bore-holes because, as earlier stated, they have been believed to represent or suggest the composition of the sea in remote geological times. The late Dr. T. Sterry Hunt wrote in the years 1860 to 1880 extensively of underground-water from this point of view. As a member of the Canadian Geological Survey he had analyzed many samples from the Palæozoic strata of Canada. He remarked that “only one-half the chlorine is combined with sodium; the remainder exists as chlorides of calcium and magnesium; the former predominating, while sulphates are present in only small amount. If now we compare this composition, which may be regarded as representing that of the Palæozoic sea, with that of the modern ocean, we find that the chloride of calcium has been in great part replaced by common salt, a process involving the intervention of carbonate of soda, and the formation of carbonate of lime. The amount of magnesia in the sea, although diminished by the formation of dolomite and magnesite, is now many times greater than that of lime; for as long as chloride of calcium remains in the water, the magnesium salts are not precipitated by carbonate of soda.” This quotation is not mentioned with a view to discussing the composition of the ancient

ocean, but as an illustration of an interesting interpretation of connate waters. Undoubtedly they are, as a rule, modified somewhat by infiltrating surface-waters, but their large features are significant. As earlier stated, the name connate was coined in 1909, long after Dr. Hunt's time, by A. C. Lane,⁸ an active member of the Institute, who has given special study to the composition of underground waters in later years. Dr. Lane also emphasizes the relatively large percentage of calcium chloride around the Lake Superior basin as soon as we draw waters from depths of from 600 to 1,600 ft., but he makes the further point that in rain-waters,⁹ which fall and circulate on the surface, or which percolate to limited depths, chlorine is present in very small amounts, provided there are no salt deposits or brines in the vicinity. A few parts per million of chlorine raise at once the suspicion of contamination from sewage. As soon as we reach the deeper waters sodium and calcium chlorides are in evidence, the former preponderating at first, but with greater depths yielding to the latter.

These characters of the deep and superficial waters are not without their useful applications. All the members of the Institute may not have heard of the case at the Vulcan iron mine, Michigan, in which at the 15th level a flow of water of 2,300 gal. per min. was struck. The question then arose whether this water was derived from a pond on the surface which might all be pumped dry by way of the 15th level, or whether the water came from an underground reservoir, and could be exhausted without tapping the surface supply. Neglecting non-essentials, an analysis of the underground water gave:

	Parts Per Million.
Total solids	340
Chlorine.....	61
Calcium.....	60
Sodium... ..	None.

The surface-water contained but 3.8 parts of chlorine per million, with high alkalies. Dr. Lane pronounced the former an underground water not indicating connection with the sur-

⁸ Mine Waters and Their Field Assay. *Bulletin of the Geological Society of America*, vol. xix., p. 503 (1907).

⁹ These are called pluvial waters by Dr. Lane.

face, and subsequent pumping exhausted it without affecting the pond.

Some sediments apparently lack free connate waters. Thus in the three extremely deep borings, from 4,500 to more than 6,000 ft. in depth, which have been put down in western Pennsylvania, and the neighboring parts of West Virginia, thousands of feet of shales and sandstones have been cut which are apparently dry. The upper waters having been cased off a few hundred feet down, water had to be poured in, in order to use the drill and the sand-pump. The deepest of all these boreholes is now being drilled at MacDonald, Pa., with the intention of making it the record hole of all those hitherto attempted. In December last, as reported by I. C. White, it had reached 6,052 feet.

3. *Magmatic Waters.*—The last variety of ground-water, the magmatic, has been in later years brought into prominence, although its existence has undoubtedly been believed in by those who have studied volcanic phenomena for many years past. We ordinarily date the application of volcanic agencies to the explanation of ore deposits from a famous paper of Élie de Beaumont in 1847.¹⁰ Interest lapsed, however, for nearly 50 years, and in general our reasoning was carried out along the lines of the meteoric ground-water in connection with the normal increase of heat with depth, and local supplies of heat from intrusive masses of igneous rocks. Gradually, however, the study of contact-zones, of pegmatites, of apatite deposits bordering on igneous rocks, of tin deposits, and of the chemical composition of igneous rocks, especially pitchstones and pearlites, with their high percentages of water, have all directed attention to the possible emission of water from molten masses of rock. These geological phenomena are all of decided and unmistakable significance. Except in the case of the pitchstones, which have from 5 to 10 per cent. of water, and the pearlites, with from 2 to 3, all the ones cited are connected with intrusive rocks and deep-seated phenomena. They have led us to believe that the molten masses of deep-seated origin held in their substance, and as much a normal part of them as silica and the common bases, water—or rather dissociated oxygen and

¹⁰ Note sur des emanations volcanique et métallifères *Bulletin de la Société Géologique de France*, Second Series, vol. iv., p. 1249 (1847).

hydrogen—fluorine, chlorine, carbon, sulphur, and some rarer elements, which, as the mass cooled and resolved itself into anhydrous, crystallized minerals, were evicted, except in so far as in relatively small amounts they became inclosed or combined in the rock-components. We can see them sometimes under the microscope in tiny cavities in the quartz of granites, and less often in other minerals. From fresh samples of rocks it is possible to drive off and collect water-gas, carbon dioxide, and minor associates, so that their presence has been both quantitatively and qualitatively determined. We find also that these elements enter into the composition of a number of minerals of the igneous rocks. Micas very commonly contain hydrogen and fluorine. Hornblende has been thought to differ from pyroxene, in part at least, because it has as an essential component a half of one per cent. or less of water (or hydroxyl, HO). Apatite, which rivals magnetite in being quite without exception in every igneous rock, contains either chlorine or fluorine. The sodalite group have in some cases chlorine, in others the sulphuric acid radical.

When the crystallization of the molten mass, with the production of anhydrous silicates, drives the larger part of the gases away, we infer that the gases carry with them other elements, such as silicon, iron, and aluminum for the contact zones; silicon with base and precious metals for the veins; and that they furnish, together with involved portions of meteoric ground-waters, our hot springs.

The most tangible evidence of the magmatic waters, we have naturally felt to be the active volcanoes. If these centers of the emission of lava and vapors can be shown to contain them, we would practically have their existence proven. Volatilized minerals which have come from just such emissions are familiar features of many volcanic vents: ammonium chloride, or salmiak; sodium chloride; boracic acid, or sassolite; sodium borate, or tinkal; specular hematite produced by the reaction of volatilized ferric chloride and the oxygen of the air; ferric chloride itself, often taken by visitors at the cone of Vesuvius for native sulphur; native sulphur; hydrogen sulphide; sulphur dioxide; carbon dioxide in the expiring stages; and, most abundant of all according to the great majority of analysts of emitted gases, water-vapor or its constituents. All

will recall the early views, based on the distribution of volcanoes along the seacoast, that the water and the chlorides of one sort or another, perhaps also the sulphur compounds, had been derived from sea-water. The sea-water was imagined to have percolated downward, and inward from the ocean until it had become involved ultimately with the masses of molten rock and with them had come to the upper world. The explosive forces developed in volcanic outbreaks were referred to these very contributions of steam. In the last five or six years Prof. T. J. J. See, of the Naval Observatory, Mare Island, Cal., has contributed several memoirs¹¹ through the publications of the American Philosophical Society, of Philadelphia, on the percolation of oceanic waters from the bottoms of the abysses of the ocean, under the pressure of the enormous overlying column of water, into the rocky crust of the earth. Professor See attributes great geologic importance to waters involved in this manner in the deep-lying strata.

Of course these questions can scarcely be solved by observation, and are unavoidably speculative in their nature. To many of us who have reflected on the matter it has seemed that water from the sea or from any other source could not percolate towards masses of molten rock at temperatures of 2,500° F. and above, and ever reach the magmas. The influence of the heat would be to drive away from the magma all such percolations, rather than to permit their approach.

The case is not essentially changed if we imagine a mass of molten rock penetrating strata supplied with connate sea-water or fresh water. The most natural effect would be to drive outward and before it a widening zone of the waters first changed to steam, perhaps ultimately to dissociated gases. In so far as the steam or gases could escape by fissures in the rock, we would naturally infer that they would. If, however, some were trapped and unable to escape and were overtaken by the advancing magma, so that their containing wall-rock was melted into it, there would be absorption. The conditions are very much like those prevailing in the woods when a noisy party of people crash through the shrubbery. A widening fan of

¹¹ New theory of earthquakes and mountain formation, as illustrated by processes now at work in the depths of the sea. *Proceedings of the American Philosophical Society*, vol. xlv., pp. 369 to 416 (1907); vol. xlvii., pp. 157 to 275 (1908).

startled and fleeing wild animals, large and small, radiates outward ahead of them. A creature in a trap cannot escape, and if edible is ultimately involved by the medium of heat in the tissues of the party. Indeed, we often speak of the digestion and absorption of wall-rocks by advancing igneous magmas, as a reader will find by looking through the extremely suggestive and able papers of R. A. Daly ¹² on the stoping of a passage by igneous magmas. Dr. Daly's conception of the mechanics of the advance of a great batholith or intrusive mass is that it breaks down the overlying rock, passes the fragments back and absorbs them in its substance; rather than that it splits the walls apart or hoists them aloft so as to make a space for itself. The former process is now very generally described among geologists by the good old miner's term of stoping.

If there are stoping and absorption by fusion of wet masses into the substance of the magma, there is a possibility of adding water-gases by a process suggested first, so far as known to the writer, by Dr. Daly. There are rocks, such as serpentine, chlorite-schist, and clays with abundant kaolin, whose minerals contain combined water in such strong bonds of affinity that only very exalted temperatures suffice to free it. If, therefore, a great uprising mass of superheated molten rock absorbed into itself serpentines, chlorite-schists, or clays, much water or water-gases might be involved with them and in such relations as to prevent the escape of the gases before absorption.

A number of geologists have been on the watch to detect in the eroded and exposed portions of batholiths the suspended or floating fragments of the wall-rocks which in the enormous stoping process involved might be expected in relatively large numbers and in all processes of absorption. We do not often have exposed just the upper portion or advance guard of the batholith and we may not perhaps attach too great weight to the observed cases yet described; but the conclusion thus far seems to be well grounded that we do not find as many stoped fragments as we would expect. Some are, of course, known. In granite quarries we sometimes see the included slabs of gneisses or schists that form the wall-rock of the intrusion.

¹² The Mechanics of Igneous Intrusion. *American Journal of Science*, Fourth Series, vol. xv., pp. 269 to 298 (1903); vol. xvi., pp. 107 to 126 (1903); vol. xxvi., pp. 17 to 50 (1908).

Yet they are sharp along their edges and often thin and ragged in their shape. Great masses of limestone have been observed in the midst of uprising batholiths or laccoliths, but they are usually changed to contact zones of garnet or similar minerals on the borders, and otherwise present a well-defined contact against the igneous rock. We appear not to find a gradual chemical or transitional passage of rock types—for instance, from a demonstrably siliceous magma, through a more basic stage to unchanged wall-rock or included fragment, such as limestone; but a sharply-cut contact between the two. Such increase in basic character as we sometimes do find from center to border in dikes seems due rather to the migration of the bases towards the borders in crystallizing than to the absorption of foreign matter. The increasing basic character is as often seen where sandstones, which are nearly pure silica, form the walls, as where the wall-rocks are higher in bases than the original eruptive. In a word, I think I am justified in saying that geologists generally admit the possibility of the fusing in of hydrated rocks and the addition of water-gases to molten masses in this way, but they have been hitherto somewhat disappointed in checking up the process by observations in the field.

Almost if not quite the only important case of the fusing into a magma of the neighboring wall-rock, so as to yield a border zone of intermediate composition, has been described from Pigeon Point, Minn., by W. S. Bayley. Between a gabbro and a slate lies a zone of syenitic rock, which Dr. Bayley believes to have been formed by the absorption of the slate by the gabbro.¹³ Authentic instances are so rare, however, as to make us inclined to fall back on the belief that the gaseous components of water; the carbon dioxide or its dissociated elements; the chlorine and fluorine; the boron; and, most important of all in connection with ore deposits, the sulphur, were all in the original magmas, as much a part of them as any other of their characteristic elements.

The discussion of the gaseous components of the molten magmas and of their possible methods of derivation has carried us a little aside from the question earlier propounded, regarding the actual demonstration of their presence amid the emis-

¹³ The Eruptive and Sedimentary Rocks on Pigeon Point, Minn., and Their Contact Phenomena. *Bulletin No. 109, U. S. Geological Survey* (1893).

sions of active volcanoes and as part of the expiring manifestations. A goodly number of investigators have been busied with this problem in the past. As well as the apparatus at command has permitted, samples of gases have been collected and have then been analyzed in the laboratory. In early years, Boussingault, Bunsen, Sainte-Claire Deville, Fouqué, and others gave special attention to them.

Some five years ago Dr. F. C. Lincoln,¹⁴ at the time a student with the writer, compiled all the accessible published analyses of these gases and tabulated them. It was then possible to note which were most abundant and which least. The leader of all proved to be water or water-vapor. Aside from it, the gases are, in order, nitrogen, carbon dioxide, free oxygen, hydrogen sulphide, hydrochloric acid, free hydrogen, sulphur dioxide, marsh-gas, carbon monoxide, and ethane.

Soon after, the researches of Dr. R. T. Chamberlin under the auspices of the Carnegie Institution were published. Dr. Chamberlin attacked the problem of the gases still remaining in the chilled and solidified igneous rocks. To the results of earlier investigators he added a great number of new determinations.¹⁵ The fact that some gases remained in the consolidated rock gave us reason to think that still more had escaped, and to this degree Dr. Chamberlin's paper afforded support to the belief in their existence.

Several years ago M. Paul Brun, of Geneva, Switzerland, determined to devote himself to the investigation of the existence of magmatic waters.¹⁶ Mr. Brun first experimented with the determination of the freezing points of feldspar and other minerals which melt at high temperatures. He then, with collecting apparatus, gave special attention to the gases at Vesuvius and at Kilauea. His results were quite startling. The vaporous cloud which hung over Vesuvius, and which had been considered steam, was referred to fine particles of ammonium chloride and other salts. The analyses of gases taken at this active center were believed to show no water, or, if water was present, it was referred to infiltrated meteoric waters, of source in the

¹⁴ Magmatic Emanations. *Economic Geology*, vol. ii., pp. 258 to 274 (1907).

¹⁵ The Gases in Rocks. Carnegie Institution (Washington, 1908).

¹⁶ Recherches sur l'exhalaison volcanique (Geneva, 1911). See especially pp. 248, 249.

immediate vicinity. Mr. Brun also visited Kilauea and took samples of gases from the crevices around the lake of lava, and from the cloud of vapor that at times flows away from the boiling lava itself. He reported no water in either, and developed the further points that from the supposed watery vapor no rainbows were produced by the sun; and that as the jets of vapor burst outward into the air there was no transparent portion, as we see in a steam-jet from a locomotive or other boiler, just as the heated vapor emerges, and before it condenses. Mr. Brun's researches excited great interest, and led to much sober thinking on the part of those geologists who had placed their faith in the existence of the magmatic waters. The case for the latter had been so well established, and the analytical results of previous chemists had been so uniformly in favor of the presence of water, that confirmation of Mr. Brun's conclusions was emphatically needed.

From another source a contribution had come bearing on this question. The vicinity of Christiania, Norway, is one of the most famous of the world's localities in the estimation of geologists. A wonderfully complete section of Palæozoic strata from the Cambrian into the Devonian and prolific in fossils has given it fame among the stratigraphers; and an even more wonderful development of rare types of igneous rocks, in batholiths, laccoliths, intrusive sheets, dikes, and surface flows, has attracted for several generations geologists from all parts of the world. In the last 25 years both the stratigraphical and the petrographical work of Prof. W. C. Brögger has served to solve not alone questions of stratigraphy, but of the variations in magmas, the derivation of different types of igneous rocks from a parent mass, and many other fundamental questions. Members of the Institute who have worked in rock-collections in their student days will recall the nephelite-syenites, the rhomben-porphyrries, and the contact zones of this region. To a few favored geologists, among them the writer, it has been given to go over the ground with Professor Brögger, and catch his contagious enthusiasm. Members of the Institute will also recall the studies in later years of Prof. J. H. L. Vogt, until recently in Christiania, but now in the new technical school at Trondjem. Professor Vogt has given much attention to the ore deposits which are found in association with the igneous

intrusives, and which indeed made the study of contact zones one of the earliest lines of investigation carried out in this region. Several years ago the detailed study of the zones was undertaken by V. N. Goldschmidt, a student with Professors Brögger and Vogt, as a subject for his doctor's degree. A very elaborate monograph has resulted, one of the most detailed ever published upon these themes.¹⁷ There are contact zones containing lime-iron garnets, specular hematite, magnetite, and minor iron-bearing minerals, and the question arose, as it has in the study of other regions, as to the introduction of the iron. Previous observers, notably Lindgren, Leith, and the writer, have invoked the agency of water or its dissociated gases as one of the chief instruments. Possible chlorides had not been overlooked, but steam or some corresponding gases at high temperatures were much respected. Dr. Goldschmidt, however, was inclined to reject the agency of steam and to consider ferrous and ferric chlorides as the only necessary compounds of iron involved. The chlorine freed by the combination of the iron with oxygen in specular hematite or magnetite was believed to find a resting-place in the mineral scapolite, a calcium-aluminum silicate, with a notable percentage of chlorine in its composition. Dr. Goldschmidt's conclusions coming at about the same time as those of Mr. Brun again gave pause to too-confident speculations regarding magmatic waters.

Still more recently, as the members of the Institute are doubtless aware, a station has been established in the Hawaiian Islands for the study of volcanic phenomena. Professor Jaggar, lately of the Massachusetts Institute of Technology, has gone to the Islands for a stay of five years. With him has been associated F. A. Perret, an American, formerly an electrical engineer in active practice, but more recently one who has made important contributions to our knowledge of volcanoes and earthquakes from his studies at Vesuvius and Etna. The past season A. P. Day and E. D. Shepherd, of the Carnegie Geophysical Laboratory, went to Kilauea equipped with the best possible apparatus for the study of the emissions and for taking samples of the gases for analysis. At some personal risk they established a suction-pipe which entered the side of a

¹⁷ *Die Kontaktmetamorphose im Kristianiagebiet*. Christiania Academy of Science (1911).

little blister-cone a short distance from the main lake of lava, and exhausted from the immediate surface of a mass of molten rock the gases which were being given off and which above the tube became ignited as flames. From these gases the two investigators, at a safe distance, were able to condense in tubes unmistakable samples of water. Dr. Day exhibited at the recent annual meeting of the Geological Society of America in New Haven a sealed glass tube about 2 ft. long and an inch or more in diameter in which was about a pint of actual magmatic water, condensed at Kilauea. We cannot but conclude that magmatic waters actually do exist, and that even from basic lavas, the least favorable to them of our varieties of igneous rock, they are delivered into the atmosphere. So far as one can see, we are fully justified in the conclusion that deep-seated magmas likewise contain them, and that when such rocks crystallize to anhydrous silicates the waters must be driven off into subterranean circulations. It is not a far step to the further conclusion that such waters bringing direct from the magmas their burdens of ores and gangue deposit the vein-fillings and impregnate the wall-rocks with ores. Nor is it surprising that we find the writers of the last 20 years, Lindgren, Vogt, Spurr, and others, coming more and more towards this point of view in explaining the widespread association of ore bodies with intrusive rocks.

The Lower Limit of the Ground-Water.—We have now before us the three kinds of ground-waters: the meteoric, the connate, and the magmatic. There is one interesting phase of the subject, which especially relates to the first two. At the outset of this paper a brief outline was given of the experience which we have gained by sinking shafts, wells, and bore-holes from the surface. Almost everywhere water is encountered at a moderate depth. Impressed by this fact, the earlier and very natural conclusion was that the water extended indefinitely downward to the limit of possible cavities in the earth. If it ultimately reached zones of heated rock, as we would suppose, circulations would be established which would tend to bring up dissolved matter from the depths and deposit it above. This was the conception widely held and most clearly and definitely formulated in the able essay of Prof. C. R. Van Hise, published in the *Transactions* in 1900. Based on the calculations

of Prof. L. M. Hoskins on the mechanics of rocks under pressure, the conclusion was reached that cavities were welded together at about 10,000 m. below the surface. We imagined, therefore, that waters of meteoric origin percolated downward to or towards some such extreme limit as this in the firmest rocks, and that the last turning-point must be placed at these depths.

More recently the extremely able and ingenious experiments of Prof. Frank D. Adams,¹⁸ of McGill University, Montreal, Past-President of the Canadian Mining Institute, and the review of the mathematics of crushing by his colleague, Prof. L. V. King,¹⁹ have shown that cavities can exist at much greater depths than 10,000 m. Thus Professor Adams has placed cylinders of granite in his testing machine. The cylinders have had holes bored through them before insertion. They have then been subjected to a compression of 96,000 lb. per sq. in. while maintained at high temperatures 70 hours. After compression the cylinders have been taken out again with the hole unchanged. These pressures would correspond to depths of 15 miles, but the temperature at which the test was made (550° C.) was that estimated to prevail in the earth at a depth of only 11 miles. The softening of the steel jackets of the granite cylinders prevented raising the temperature above 550° C. We must conclude that cavities can still exist a long way below our former limit, and Professor King's revision of the earlier calculation has brought out the fact that the earlier conclusions were based on an assumption which appears now not to correspond with the conditions in nature. If the old belief in the downward percolation of surface-waters under the attraction of gravity is well grounded, then the depths to which the waters may go, aside from the expulsive influence of high temperatures, are greater than we used to think.

But, again, as once before, we may appeal to the experience of the miner, and in this respect members of the Institute, by recording experience, may contribute valuable data to the subject. In later years shafts have been penetrating to greater

¹⁸ An Experimental Contribution to the Question of the Depth of the Zone of Flow in the Earth's Crust. *Journal of Geology*, vol. xx., pp. 97 to 118 (1912).

¹⁹ On the Limiting Strength of Rocks under Conditions of Stress Existing in the Earth's Interior. *Journal of Geology*, vol. xx., pp. 119 to 138 (1912).

and greater depths. We are thus able to test the water-conditions by actual observations. So far as possible in mining, it is important to confine the waters to the upper levels and to save the expense of elevating them from greater depths. While hitherto this paper has dealt with the general aspects of this question and has avoided details, it will not be without interest if a few new cases of mines are added to those already on record.

If we are not able to impound the waters in the upper levels, but if at the same time with increasing depth the amount of water pumped remains the same, obviously the deeper rocks are not contributing any additional supplies. They must be relatively, if not quite, barren of free water. If we can impound the waters so as to prevent their descent, and then are not obliged to pump at all from still lower levels, again it is clear that the deeper rocks are dry. The copper mines of Keweenaw Point and the deep bore-holes of western Pennsylvania are of the latter type.

In earlier papers on this subject or on previous pages in the present contribution reference has been made to the mines at Mapimi, Durango, Mexico; Tintic and Frisco, Utah; Silver Plume, Colo.; Keweenaw Point, Mich.; Johannesburg, South Africa, and several European mines cited by Posepny. J. W. Finch has remarked the lack of water in the deep workings at Butte. On account of the quenching of mine-fires in recent years, the water-records at Butte are not easy to state with accuracy. John Gillie estimates, however, in properties not affected by water poured in for this purpose, the increase of 1,200 ft. in depth, from the 1,000-ft. level to the 2,200-ft., has only added a probable 5 per cent. increment to the water pumped. The increase has certainly not exceeded 10 per cent. From two square miles of area, as drained by a number of shafts from 2,200 to 2,800 ft. deep, the total water does not exceed 3,500 gal. per min. The mines at Butte, being on the gentle south slope of a mountain side with a height of land behind them, and a deep, gravel-filled valley around two sides of the slope below them, are in very favorable situation to gather up water. The granite is greatly fissured, but clay gouges are well-nigh universal. Under the circumstances of the huge stopes, over 2,000 miles of workings, the great depths and the

extended area, this output of water is much smaller than might have naturally been anticipated.

From William Hague, by the kindness of J. R. Finlay, the following experience at the North Star mines, Grass Valley, Cal., has been supplied. The first 1,000 vertical feet yield about 250 gal. per min. For the next 1,300 ft. the additional water is only about 100 gal. per min. A decrease with depth is thus indicated, leading one to infer a progressive falling off.

From Roger T. Pelton, of the Copper Queen Co., and through him from Mr. Gehring, of the Calumet & Arizona Co., both at Bisbee, Ariz., it has been learned that the belief prevails in the decrease of water with depth. Could the water from upper workings be cut off, the lower workings would probable be comparatively dry. Experience shows that new developments may strike water-ways which let in temporary inflows. When these are exhausted by pumping, the workings are comparatively free. The chief pumping of both companies is now done from the Junction shaft, at a present depth of 1,800 feet.

Some interesting data regarding the Telluride district have been received from Gelasio Caetani. Tunnels are generally possible in the local mines, and the water is believed to originate from infiltrations along the outcropping vein-fissures. Very little comes from the country rock. The total water is heaviest at the time of melting snows, and falls off in the winter to very small amounts. At the Camp Bird mine, however, after having had comparatively dry workings down to 500 ft. below the lowest tunnel, the miners met an inflow of water so heavy as to preclude further development. Some open channels must have led in water from a large reservoir or source of supply at a distance. The mine was still comparatively shallow, but the experience showed great irregularity of distribution. We may be reminded in this connection of the experience at Cripple Creek, Colo., which is summarized by Lindgren and Ransome in their valuable monograph on the district. The open-textured breccia in the throat of the old volcano is a vast reservoir whose waters continue to the depths thus far reached in mining. The surrounding granite is, however, comparatively dry, and yields but little water to the extensive tunnels which penetrate it. The waters which now pour from the tunnels are cold, and are naturally referred to

the melting snows and rains. The two authors conclude that the waters will continue below 2,000 ft. The enormous amounts which in the aggregate have poured from the tunnels are very impressive, but once the storage reservoir above these levels is exhausted they supply but little.

Through Mr. Caetani some figures for the Bunker Hill and Sullivan mines in Idaho have been received from Stanley A. Easton. The Kellogg tunnel, over 2 miles in length, is at the end some 1,500 ft. immediately below the top of the ridge, and 2,000 ft. or more beneath closely neighboring summits, but it is much less than this below the neighboring valleys. The overlying mountain contains the immense workings of former years, which are in part at least drained by the Reed, Sweeney, and Arizona tunnels. Yet while formerly 200 gal. per min. was supplied from this portion to the Kellogg tunnel, the water has now been practically exhausted and the supply to-day comes from the Kellogg shaft, now 800 ft. vertically below the tunnel. Experience has shown that each additional 100 ft. depth to the shaft adds 50 gal. per min. to the duty of the pumps. This rate, however, Mr. Easton thinks will not continue. The earlier experience with the tunnel would lead one to anticipate the exhaustion of the water in the strata. The rocks are nearly vertical quartzites, rather thinly bedded, and cut by a great crushed fault-zone, with well-developed gouges.

Experience has shown the sympathetic connection of bodies in wet ground for distances that are impressive. The heavy pumping-duty encountered at Tombstone, Ariz., affected mines a mile distant, and probably drew waters along a fault from a storage reservoir much farther away. The famous case of the lignite mines at Dux, Bohemia, which suddenly tapped off the hot springs at Teplitz, 4 miles away, has been cited by Posepny. The waters were plugged off so as not to destroy a health resort.

We cannot indefinitely multiply instances, some of which do not bear very closely on the matter in question, but the general experience leads us to the conclusion that the meteoric ground-waters give out in depth, nor do they penetrate certainly in important amounts in firm rocks much below 1,000 to 2,000 feet. The rocks are tighter than we have sometimes supposed.

Along another line of attack some interesting data are afforded by the conclusions, six years ago, of M. L. Fuller, formerly of the U. S. Geological Survey, and later attached to the Water Supply branch. Mr. Fuller reviewed the amount of water in the rocks forming the outer part of the earth's crust, on the basis of extended observations upon bore-holes, wells, and excavations. The amount is expressed as a sheet, spread out over the entire surface of the globe, a method adopted by several earlier writers on the subject. Quoting from Mr. Fuller's paper,²⁰ the estimates are as follows:

Delesse, 1861, sheet of water 7,500 ft. deep.

C. A. Slichter, 1902, sheet of water 3,000 to 3,500 ft. deep.

Chamberlin and Salisbury, 1903, sheet of water 800 ft. deep on one assumption.

Chamberlin and Salisbury, 1903, sheet of water 1,600 ft. deep on another assumption.

C. R. Van Hise, 1904, sheet of water 226 ft. deep.

M. L. Fuller, 1906, sheet of water 96 ft. deep.

The tendency, thus, in later years has been to greatly reduce the amount. Our inevitable inference is that the water does not descend very deeply.

How, then, are we to explain our veins which in frequent instances reach much greater depths than 1,500 ft. and are usually the remnants left by earlier erosion, so that when formed they ought to be credited with even greater vertical extent than they now display? If we deal with meteoric or free connate waters we must assume, in order to admit them to the observed depths, an extensive production of open fissures which will locally supply waters from the upper reservoirs to the lower rocks. Internal heat or the heat from local intrusive masses, which might themselves be responsible for the fissuring, would then start upward currents and conceivably produce circulations. The derivation of the ores and gangue by solution, however, from the condition of extremely sparse distribution which our assays of rocks reveal to us, and only from along well-defined fissures, is a very slow, if not an impossible, process. Percolation through the walls would not seem reasonable

²⁰ Total Amount of Free Water in the Earth's Crust. *Water-Supply and Irrigation Paper No. 160, U. S. Geological Survey*, pp. 59 to 72 (1906).

when the upper reservoir, with all the pull of gravity, will not pass downward. In fact, the limitation of our ground-waters to the upper zone or to relatively large and open fissures puts a new aspect entirely upon our reasoning regarding vein-filling. It handicaps very seriously the processes of leaching and concentration in which we formerly felt confidence. If, impressed with the difficulties, we cast about for an alternative agent, we find it in the magmatic waters and have refuge in the belief that they issue from the intrusive masses and carry ores and gangue in solution to the veins.

DISCUSSION.

N. H. DARTON, Washington, D. C.:—This question of underground water is of interest to me, for I have given much attention to artesian water problems and the capacity of various rocks to hold water. I believe that some underground water sinks very deep into the earth in porous strata and joints, even in relatively compact rocks and in arid regions. I do not refer to palpable water which runs out of mines or yields supplies to wells, but the interstitial water or dampness. I have examined many borings from deep wells and shafts which appeared to be dry, but on analysis they proved to contain several per cent. of uncombined water. In many cases the amount of water in these dry-looking materials is sufficient to sustain chemical reactions, and doubtless by capillary action serve as medium of underground movement of various dissolved substances. It is my belief that such dampness extends to very great depths and in all regions. I think we should be cautious in giving it an upper limit as we do in referring to the "water plane," and its lower limit may be much deeper than is generally supposed. It is not safe to judge the limits of underground water by its presence in noticeable amount.

THE CHAIRMAN:—Would not you think the dampness would be a very efficient agent to fill a good strong vein?

MR. DARTON:—Possibly not unless it acted for a long time; time would undoubtedly be a most important factor. Very important.

THE CHAIRMAN:—Some years ago we had in the *Transactions*, as many members may recall, a paper by H. P. Gillette, entitled, *Osmosis as a Factor in Ore-Deposition*.²⁶ The point of that paper, as I recall it, was that the circulation of waters and the introduction of new matter by way of convection may not be necessary in vein formation, but that, in a standing and quiet body of water saturating the wall-rocks of a fissure, the component parts of the gangue may move in a procession of ions to centers of crystallization. If centers of crystallization were once started, the respective ions would be drawn towards them and would accumulate in this way. They would march through the medium of solution itself without convection currents. It is possible that in this way metals may have been concentrated from large bodies of rock; but most of our strong veins, certainly those large enough to mine, have seemed to us to have required circulating waters, which brought in the ores in solution and served as a vehicle of transportation. I must say I do find some difficulty in imagining that the dampness which stands on the walls as small beads of water, analogous to perspiration, is sufficient to produce a very powerful vein. By the process which Mr. Gillette has described, widely dispersed metals could be collected from very solid walls. In the recrystallization of matter to form minerals in thin, flat forms, as in the mica schists, there is a transfer of the mineral components, with the aid of small quantities of water, so as to become mica, in a position at right angles with the pressure. Very little water is needed for the transference to the new position. The production of concretions in shale and of the "clay dogs" which we have in clay beds has been due to the circulation of water along sandy layers and the bringing of calcium carbonate in solution until it met some nucleus around which it was deposited.

In the case of the flints in England, I should imagine that water acted as a vehicle for the collection of the disseminated silica. The masses of flint often have some connection with an old sponge or some other center of precipitation. Most of our chert and flint in America seems, however, to have passed through a gelatinous stage to its present hard condition.

²⁶ *Trans.*, xxxiv., 710 (1903).

The Sulphide Ores of Copper. Some Results of Microscopic Study.

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CAMBRIDGE, MASS.

(New York Meeting, February, 1913)

TABLE OF CONTENTS.

	PAGE.
I. INTRODUCTION	26
1. The Relations of Scale in Geologic Work	26
2. General Characteristics of Copper Sulphide Ores.	29
3. Scope of Present Study	31
II. DESCRIPTION OF THE IMPORTANT MINERALS	36
1. Pyrite	36
2. Pyrrhotite	41
3. Chalcopyrite	41
4. Bornite	45
5. Covellite	50
6. Chalcocite	52
7. Enargite	59
8. Tetrahedrite and Tennantite	61
III. CONCLUSIONS	62

I. INTRODUCTION.

The Relations of Scale in Geologic Work.

MANY features of human accomplishment rest upon the possibility of representing natural objects on a scale of different magnitude from the actual one. Most drawings, for example, are of very different scale from that of the object depicted, and indeed are useful mainly on this account. More important than the question of mere convenience, however, is that of meeting the physical limitations of vision.¹ Thus, a mural

* Non-Member.

¹ The field of distinct vision in the normal human eye is surprisingly small, being ascribed an angular value of only from 2° to 4°; the area of most acute sensibility, which alone is effective for careful visual examination, is still smaller, corresponding to an angle of only one-tenth of this magnitude. Moreover, the limit in the other direction is quickly reached, since the average eye records only a single image when the angle of vision between two objects is less than 0°1', and accurate impressions of form and color probably fail before even this degree of diminution is reached. It thus appears that the eye, although so useful and effective an organ to us, possesses as an optical instrument a very narrow range.

painting may possess a beauty of proportion which could hardly have been attained except by a small preliminary sketch in which the relations of the whole could be comprehended at a glance; on the other hand, an innocent-looking drop of water may become a crowded storehouse of potential disease when examined on the scale applied by the bacteriologist.

This adaptation of scale to requirements is nowhere more common or essential than in geologic work. The topographic and geologic maps not only bring the features of large areas under convenient observation but facilitate deductions that in some instances might otherwise be impossible. For instance, the correlation or extension of interrupted exposures of a vein or fault, possibly separated by intervening hills, may be difficult on the ground but simple when condensed within a single field of vision on the map; similarly, vertical cross-sections afford conclusions as to rock structure that might never be possible without reduction of the natural scale.

Equally common and even more necessary in geologic work is increase of scale—the pocket lens is as much the geologist's badge of office as is the hammer or the compass. Greater and more accurate magnification has been demanded by the modern developments of the science, and in at least one branch, petrology, has been afforded by the petrographic microscope—that highly specialized instrument whose constantly increasing employment during the last half-century has done so much to advance and perfect geologic conceptions. At the first, unaltered igneous rocks seemed to offer the most attractive and fruitful field for microscopic investigation, and unfortunately the average petrographer of the present day more or less unconsciously still confines his attention chiefly to such geologic products. Even in the early days, however, a few geologists realized the importance of microscopic investigation of secondary products, particularly ore deposits and the surrounding altered country-rocks. The systematic geological study of ore deposits, emphasized especially within the last 15 years, at a period of high development of microscopic investigation, has naturally relied increasingly upon the microscope for solution of many of the problems encountered. Pioneer work in this direction by Stelzner in Germany and by Pumpelly on the Michigan copper deposits, has in recent years been carried much further, espe-

cially by Professor Lindgren, and has led us to vast funds of knowledge concerning the deposition and alteration of ores.

Though possessing a high degree of perfection for the examination of that preponderant class of minerals which in thin section are transparent, the petrographic microscope, nevertheless, fails in one important particular when applied to the examination of ores. Because the instrument is designed to employ transmitted light, whereas the majority of ore minerals are opaque, most of the optical properties depended on for petrographic diagnosis are not available and only low magnification and inadequate illumination can be secured. Satisfactory microscopic investigation of opaque minerals and ores has been possible only since Professor Campbell² a few years ago conceived the idea of applying to their study the method long in use by metallographers of examining a polished face of the material through a special microscope that directs light upon this polished surface, from which it is reflected perpendicularly back to the eye of the observer. By this means it is possible to control illumination and magnification within wide limits. The relations thus disclosed in mineral aggregates, when interpreted in the light of their geologic occurrence, are wonderfully instructive and throw much light on the processes of ore deposition and alteration. That a metallurgist should apply this metallurgical instrument outside of his own field has indeed proved fortunate, and geologists are greatly indebted to Professor Campbell for affording them so remarkably effective a method of research—a method whose possibilities are yet unfathomed, and whose importance cannot be over-estimated.

As recently pointed out by so high an authority as Professor Beck,³ of Freiberg, the thorough microscopic examination of ores is of no less importance commercially than scientifically; and in this country at least there already exist indications that at no distant date the first-rate mining companies will no more think of directing exploration without advice of a competent microscopist than they now would endeavor to smelt ores without the services of an assayer, or to erect a mill without an experienced designer.

² Campbell, William, *The Microscopic Examination of Opaque Minerals*, *Economic Geology*, vol. i, No. 8, pp. 751-756 (Sept.-Oct., 1906).

³ Beck, R., *Über die Bedeutung der Mikroskopie für die Lagerstättenlehre* (Freiberg, 1911).

2. General Characteristics of Copper Sulphide Ores.

Most copper deposits which now exist in the condition of initial deposition, and are therefore regarded as primary,⁴ contain the metal in the form of sulphides, or of closely related sulph-arsenites, sulph-antimonites or similar compounds. (For convenience these various compounds of copper with the metalloids will hereafter in this paper be referred to under the general name of sulphides, except where specific mention is made of their composition.) The magmatic segregations, the pegmatitic and contact-metamorphic groups, the veins, both of the deep-seated and the shallow-seated types—all these genetic classes of deposits carry copper originally in the sulphide form; even disseminated deposits in sedimentary rocks, of the type represented, for instance, in the Colorado Plateau region, are indicated by a growing volume of evidence to have been probably in all cases originally sulphide deposits. The one important exception to this general rule of primary occurrence in sulphide form is found in the native-copper deposits, such as those of Michigan; these, it must be confessed, are as yet by no means understood.

The early realization that the oxide, carbonate, and silicate compounds, and in some cases metallic copper itself, are derivatives of the sulphides under the influence of oxidation near the surface constituted a most important advance both in its scientific and its practical aspects. Perhaps of even greater significance was the recent almost simultaneous and mainly independent recognition by several observers, including Douglas, Smyth, Emmons, Weed, and Van Hise, that some of the sulphides are themselves likewise derived from the original

⁴ The terms "primary" and "secondary" as applied to minerals or ores are objectionable because they have been used to convey conceptions both of sequence and of genesis. Epigenetic ore bodies are secondary with respect to the wall rocks, but may be primary *as ores*, in the sense that they have undergone no change since first deposited. Similarly, chalcopyrite in an ore may have crystallized subsequent to pyrite, yet may be primary in that it was derived from the same solutions as the pyrite, both minerals being initial constituents of the ore body. In the absence of satisfactory terms for either the time or the origin conception, so that "primary" and "secondary" may be limited to precise meaning, they must be employed for the present and ambiguity avoided by definition. In this paper, minerals called primary are regarded as products of original deposition from solutions probably ascending, heated, and of alkaline character; secondary sulphide minerals are those produced from other sulphides by the action of descending, cold, acid (or possibly neutral) solutions.

sulphide minerals by the influences active at shallow depths. The importance of this process of secondary sulphide enrichment may be forcibly indicated by the fact that fully 75 per cent. of the production of this country and probably not far from that proportion of the total world's output is now derived from ores which have been more or less affected by it. In spite of the attention which the importance of this subject has directed towards its study, much remains to be learned concerning it,—even, as Winchell⁵ and Sales⁶ have recently indicated, the fundamental requisite of positively distinguishing between the primary and the secondary sulphide minerals,—and in consequence many of our conclusions regarding the application of secondary enrichment are still open to doubt.

The complex mineralogy of copper — more varied than that of any other metal, and exemplified as well in the compounds with sulphur and related elements as in the oxygen- and halogen-bearing compounds — permits very slight changes in the chemical character or physical condition of the solutions from which the copper minerals were primarily derived to be expressed by different mineral varieties, and allows numerous minerals of still different composition to develop when these initial products are acted upon by later solutions of different nature. The minerals of a given copper occurrence afford, therefore, a uniquely significant and sensitive index of the conditions under which it was originally formed and of the changes which it has since undergone. It is thus evident that detailed investigation of copper mineralogy from the geologic standpoint, obviously desirable on account of the knowledge it may give of copper deposits themselves, is of far greater importance because its possibilities of shedding definite light on the whole problem of ore deposition are probably unexcelled by any other single line of geologic attack. That the results already accomplished by this means are not nearly commensurate with the possibilities apparently rests in part on failure to sense the opportunity fully, but it is also dependent on the fact that many of the mineral relations are so intimate and developed on such a small

⁵ Winchell, H. V., Secondary Sulphide Enrichment, *Engineering and Mining Journal*, vol. xciii., No. 7, p. 367 (Feb. 17, 1912).

⁶ Sales, R. H., Review of Butte Geological Report, *Engineering and Mining Journal*, vol. xciv., No. 16, p. 730 (Oct. 19, 1912).

scale that their adequate comprehension has been possible only since the application of the metallographic microscope to the study of ores. From this time forward, advances in this field should be rapid.

3. *Scope of Present Study.*

Having in mind something of the desirability of further knowledge and the possible means of attaining it, as outlined in the preceding pages, and encouraged by the results gained by certain Harvard students working with the metallographic microscope during the two previous years, we set out in 1911 on a systematic microscopic study of the sulphide copper ores.

Our primary object was to learn as much as possible of the general features of these ores: the habit, composition, stability and alteration of each of their minerals; the relations of sequence, position, and constitution which these minerals bear to one another; and finally the suggestions these findings might indicate and the conclusions they might afford. In addition to this broad, abstract object of attack, we had at the outset three specific points of inquiry, parts of the main problem. These were: (1) In pyritic copper ores, especially in those of low grade in which the presence of copper is evidenced only by assay, does the pyrite contain copper as an essential part of its constitution, possibly replacing a portion of the iron as magnesia replaces part of the lime of calcite to form dolomite; or is the copper content accounted for by definite copper minerals existing in small particles included within or between the grains of the predominant pyrite? In other words, is there strictly such a thing as "cupriferous pyrite"? (2) In certain pyrite copper ore bodies in schistose rocks, is the granular and crushed appearance of the pyrite and the enveloping mesh or cementing structure of the chalcopyrite due, as stated by Professor Emmons,⁷ to regional metamorphism and actual external deformation at a depth that is virtually the zone of flow for the chalcopyrite but the zone of crushing for the more rigid pyrite; or, as many thin sections from other types of deposits had indicated, is this relation inherent, and characteristic of pyritic copper ores? (3) What are the criteria, if such exist, for distinguishing primary copper sulphide minerals from those de-

⁷ Emmons, W. H., Some Metamorphosed Ore Deposits and the so-called Segregated Veins, *Economic Geology*, vol. iv., No. 8, pp. 760, 766, 779, 780 (Dec., 1909)

veloped secondarily; in short, how can primary and secondary copper ores be positively identified?

The first lot of material for study consisted of typical and critical specimens selected from collections made by one of us from most of the important copper districts of this country. Since these collections of ores and related wall-rocks had already been studied in some detail, mostly in thin section,⁸ we thus possessed, as a check upon our microscopic findings, knowledge of the occurrence and genetic significance of the specimens of this group, which comprises about one-half of the total number we have as yet studied. The greater number of the remaining specimens consisted of suites of various sizes from many localities, mostly collected especially for this purpose by geologists, engineers, or mine-owners, who furnished also more or less descriptive data, so that in most of these cases, also, we were able to compare microscopic indications with facts of occurrence. A very small part of the material we have studied consists of mere "specimens," taken from the college collections or sent to us from various sources, and though perhaps accompanied only by a label giving time and place of collection, we have studied them because of certain interesting features which they gave promise of revealing. The specimens studied, numbering now almost 500, have been drawn from some 75 different districts of four continents.

It is felt, therefore, that we have worked mainly on live, representative material, the geologic significance of which has been pretty well known to us, and in consequence we are inclined to regard our results with much more assurance, even though most of the microscopic relations are evident and require no corroboration. In the case of conflicting evidence, however,

⁸ Thus far we have failed to secure satisfactory results from the examination of thin sections with vertical illumination, and have been obliged to study the gangue minerals and the opaque minerals in different pieces. An expedient that is generally satisfactory, however, consists in cutting a specimen in two with a thin diamond saw, polishing the resultant face of each half, using one for opaque examination, and grinding down the rough side of the other half parallel to its polished face until the slice attains the requisite tenuity. In this way is secured a thin section from a zone distant only a fraction of a millimeter from the polished surface of the opaque piece, and in specimens of medium or coarse microscopic texture, the separated portions of the same mineral grains can be identified. We have not yet had time to pursue this combined petrographic-metallographic line-of-examination further than to conclude that its possibilities are considerable.

it has been our general rule to give precedence to that furnished by the broad field-relations rather than to the indications afforded by the microscope. This rule has been reversed in some cases where the field-evidence was doubtful and the microscopic evidence clear, and also in certain instances in which the microscope showed relations clearly parallel to two or more occurrences in which the field-evidence and that of the microscope were in accord. Finally, there remain a number of instances in which the evidence yet available is insufficient for final judgment. These cases might be regarded as discouragingly numerous were it not for the fact that they generally succumb at last under accumulating evidence from new material, which in turn may bring forth additional difficulties. It is, admittedly, somewhat disconcerting when a favored hypothesis that may have held for weeks or months is totally upset by the evidence contained in a single new specimen. But in most such cases the guilty occurrence has shown interesting new features that more than balanced any disappointment that it may have caused. For example, we worked for more than a year without finding any positive evidence of secondary bornite or more than two doubtful occurrences in which it might possibly be secondary. At last the inevitable specimen arrived containing bornite that beyond all doubt was secondary. Not only did this definite recognition of secondary bornite serve to confirm the view that the mineral in the earlier specimens was primary and to clear up the two doubtful cases, but this specimen also afforded much of significance as to the stability and nature of alteration of several of the copper minerals and is undoubtedly one of the most instructive of our collection.

Starting out with a large collection of specimens the general composition and character of which we are aware, we were able to escape in large measure a difficulty that might be expected with this new method, namely, inability to identify the minerals in the polished surface under the microscope. In the lists below are given the cupriferous and associated metalliferous minerals of opaque or sub-transparent character that we have as yet encountered and identified in the copper ores. These are in addition to such transparent varieties as azurite, malachite, anglesite, etc.

Copper-Bearing Minerals.

Algodonite,	Cu_6As .	Emplectite,	CuBiS_2 .
Bornite,	Cu_5FeS_4 ?	Enargite,	Cu_3AsS_4 .
Bournonite,	$(\text{Pb}, \text{Cu}_2)_3\text{Sb}_2\text{S}_6$.	Famatinite,	Cu_3SbS_4 .
Chalcocite,	Cu_2S .	Mohawkite,	Cu_3As (with Ni and Co).
Chalcopyrite,	CuFeS_2 .	Rickardite,	Cu_4Te_3 .
Copper,	Cu .	Stromeyerite,	$(\text{Ag}, \text{Cu})_2\text{S}$.
Covellite,	CuS .	Tennantite,	$\text{Cu}_3\text{As}_2\text{S}_7$.
Cuprite,	Cu_2O .	Tetrahedrite,	$\text{Cu}_8\text{Sb}_2\text{S}_7$.
Domeykite,	Cu_3As .	Whitneyite,	Cu_3As .

Associated Ore Minerals.

Argentite,	Ag_2S .	Pentlandite,	$(\text{Fe}, \text{Ni})\text{S}$.
Arsenopyrite,	FeAsS .	Polybasite,	Ag_3SbS_6 .
Galena,	PbS .	Pyrite,	FeS_2 .
Gold,	Au .	Pyrrhotite,	$\text{FeS}(\text{S})_x$.
Limonite,	$2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$.	Silver,	Ag .
Magnetite,	Fe_3O_4 .	Specularite,	Fe_2O_3 .
Marcasite,	FeS_2 .	Sphalerite,	ZnS .

Nevertheless, among the minerals of gray or white color especially, the designation as "unknown" became so frequent that something more than a year ago we began, in conjunction with the general work upon the copper sulphides, a study of the opaque minerals as a whole, using known specimens of all the varieties which we have been able to secure. By applying micro-chemical methods as the chief means of diagnosis, this work has now reached such a stage as to offer much promise of useful results, but it is not yet perfected to such a point as enables identification of certain mineral varieties occurring in the copper ores in microscopically small grains, and these still have to be classed as "unknown."

What is true of minerals is even more true of relations and processes; many we feel are now understood thoroughly, but there are a number of which we are doubtful and some as yet baffle us completely. The problem has expanded so remarkably with increasing understanding of certain of its phases, and the need for a large amount of further work is so evident, that the propriety of publishing this paper was seriously questioned. Certain fundamental conclusions, however, seemed to us now well enough established and of sufficient interest to justify their presentation, and the deciding argument was furnished by the hope of securing from other investigators such criticisms and

suggestions as may be of great assistance in further work of this kind and in correcting errors already made.

The work of actual microscopic examination, at once absorbing and exacting, has thus far prevented a systematic survey of the appropriate literature. Speculations upon the chemistry of the various processes exhibited are not attempted here because we feel that chemical equations, unless supported and confirmed by systematic analytical work, are as likely to be misleading as helpful. Neither have we aimed at this time to assemble by districts the features of occurrence in the various localities represented by material. We have sought merely to summarize our most important findings of fact regarding the sulphide minerals most common in the ores of copper; omitting many details of minor variation. Of the few conclusions given at the end, some will doubtless be somewhat modified by further work. This paper, then, is nothing more than a partial and tentative report of progress. In view of this condensed presentation and the inconclusive character of the main results, the introductory sections of this paper may appear elaborate and top-heavy, but are given because a reasonably comprehensive statement of the whole problem at this time seemed desirable, especially as it might serve in stimulating more effective criticism of the results herein presented.

The number of microphotographs at the end of this paper should be augmented several fold to illustrate adequately the varied relationships exhibited in the copper ores, many of which could be properly shown only in colored illustrations. The views given, however, depict several of the most common and significant features encountered.

To the many gentlemen who have kindly given us material and information, we desire to express our thanks, and we can only hope that in some possibly useful hint contained in these pages they may find partial return for their courtesy. Only the number of those who have thus favored us prevents individual acknowledgment of our obligation. We cannot fail, however, to express our indebtedness and thanks to the following, who have supplied us with large suites of specimens and valuable information concerning them: Reno H. Sales, Chief Geologist of the Anaconda Copper Mining Co.; Arthur Notman,

W. B. Gohring, Superintendent, and Ira B. Joralemon, Chief Geologist, of the Calumet & Arizona Mining Co.; and C. T. Brodrick, Geologist for several Russian and Siberian companies. We also desire to thank Prof. Albert Sauveur and G. A. Reinhardt of the Harvard Mining School, and H. M. Boylston, formerly of that Department, for our use of their metallographic laboratory and equipment, and for much valuable advice regarding microscopic manipulation.

II. DESCRIPTION OF THE IMPORTANT MINERALS.

1. *Pyrite.*

Primary Character.—Though containing no copper itself, pyrite is probably the commonest metalliferous constituent of the sulphide copper ores, and therefore deserves consideration in this connection. It is almost invariably primary, and, with few exceptions, is the oldest mineral in the ore, having commonly formed earlier than the gangue, or the other ore minerals except magnetite and specularite. The mineral occurs typically in compact and roughly equidimensional forms, which may range from perfect crystals to polygonal, sub-angular or even well-rounded grains.

Crystals of good outline are most common in those specimens in which pyrite constitutes an unimportant portion of the ore and exists in individuals not adjacent to one another. Under such conditions, since it was practically the first mineral to be precipitated, it was able to attain rather ideal molecular development. This is well exemplified in a megascopic way by some of the ore of Ducktown, Tenn., in which scattered large well-formed crystals of pyrite are imbedded in the other later sulphides in a manner that suggests the phenocrysts in a porphyry. This is also commonly shown microscopically in specimens from many localities. When the proportion of pyrite becomes greater, a polygonal outline of the grains is common, Fig. 1. These geometric boundaries are probably not usually crystallographic faces, but due rather to simultaneous growth of the various grains until they touch, paralleling the polyhedral bubbles of soap-suds in the respect that each individual grain would normally have assumed a different contour had there not been mutual interference with its neighbors. Ore in which this polygonal or mosaic structure is characteristically shown gen-

erally carries only small interstitial grains of gangue or other minerals and is therefore commonly very low in copper, present usually in the form of chalcopyrite, Fig. 1.

This polyhedral structure, however, is not characteristic of all the highly pyritic ores. Instead, the pyrite grains appear in many cases to be crushed, Fig. 22, and the fragments more or less disarranged, Figs. 2 and 3. In and around these resulting irregular but generally rounded grains of pyrite, the gangue and other later minerals of the ore have deposited as a cement or inclosing mesh. This is the characteristic structure of the average primary copper ore. The cement consists mostly of one, or an intergrowth of two or more, of the following minerals: chalcopyrite, bornite, and rarely chalcocite, with gangue and sphalerite also common. In such ores there can be no doubt that the pyrite has undergone actual mechanical deformation, for in some instances it is found that somewhat separated fragments would fit perfectly together, even to the minor irregularities, and would collectively form a compact grain if the intervening cement were removed. Such an occurrence gives the appearance of an exploding bomb. The cause of this crushing and distortion is not yet clear, but it is plain that it took place in an early stage of crystallization of the ore, because no trace of such a thing is commonly to be found in the later sulphides or in the gangue, in the latter of which it would be expected that crushing or at least strain-shadows would be seen in thin section if these materials had been subjected to stress. The fractures traversing the pyrite may be irregular and branching, but more commonly they are sharp and fairly straight, sometimes arranged in such a regular way as to suggest incipient cleavage or imperfect parting. These fractures, which are ordinarily confined to the limits of a single crystalline individual, are to be distinguished from more orderly parallel breaks of one or more systems, developed probably by external forces long after the ore was completely formed. In these the directions of fracture persist over areas much greater than those of individual pyrite grains and not uncommonly can be correlated with major structural features, such as faults or sheeting. The significant difference in these two kinds of fracturing is shown in the contrast between Fig. 3 and Figs. 4, 5, 6, in the first of which the fracturing may be regarded as autogenous

and earlier than the cementing chalcopyrite, whereas in the three figures last named, the fracturing, emphasized and exaggerated in appearance by partial alteration of pyrite, has resulted from external causes subsequent to ore formation.

This autogenous type of crushing in the pyrite and the cementing nature of the cupriferous sulphides are characteristic not only of deposits in regionally metamorphosed country-rocks, but of lodes in massive igneous rock and of contact-metamorphic deposits in limestones, in the latter case, even where the bedding is still well preserved and no evidence of mechanical deformation exists. The specimen illustrated in Fig. 2, from the Copper Queen mine, Bisbee, is an example of this last-named condition.

In addition to their occurrence as interstitial or inclosing cement to pyrite, the minerals of later formation, especially the copper-bearing varieties, occur rather commonly within individual grains of pyrite as included drops or blebs. This rounded form is especially characteristic of the metalliferous minerals, for the gangue inclusions are generally of more irregular or ragged outline. Rarely, as is well shown in a specimen from Kyshtim, Russia, these inclusions have an angular shape that suggests possibility that their form was determined by crystal-planes of the pyrite individual within its interior, *i.e.*, negative crystals. These inclusions are not to be confused with those cases in which a concave surface of a pyrite grain, truncated by the polished surface, gives the appearance of wholly surrounding the mineral that really only projects into this concavity. These included blebs consist commonly of one or more of the following minerals: chalcopyrite, bornite, chalcocite, sphalerite; but when still other minerals, such as enargite or tetrahedrite, are plentiful in the ore, inclusions of these may also be found in the pyrite. The true significance of these inclusions is not known, but it seems likely that they represent small portions of the mother liquor from which the pyrite was crystallizing, caught up in liquid form within the growing grain. If this be true, they indicate the presence, throughout the process of precipitation, of the materials that crystallized during the later stages, in contrast with the idea held by some that changing nature of precipitate indicates a corresponding change in the composition of the solution.

*Alteration.*⁹—Pyrite is decomposed under the conditions of secondary enrichment, and is converted into and replaced by other sulphides, most commonly by chalcocite, but also by chalcopyrite, and perhaps, in some cases, by bornite and by covellite. In the average copper ore, however, pyrite is one of the last minerals, except chalcocite, to succumb to such alteration, and residual cores of it may remain when nearly or quite all the other minerals have been converted to the final product, which appears to be chalcocite. When undergoing this replacement, pyrite never exhibits the slightest trace of an external zone transitional to the encroaching mineral, but retains a bright, clean appearance and clear-cut, generally angular outline even when original grains have been reduced to mere residual specks. As will be seen later, this behavior is in striking contrast with that often seen in the case of chalcopyrite and bornite. Like a thoroughly seasoned army, pyrite, even in retreat, presents a solid front to the invading foe, and thus conceals the fact that it is about to be annihilated. Consequently, recognition of replacement of pyrite by the cupriferous minerals rests in large part on the shape and disposition of the pyrite grains. Figs. 4 to 6, which represent, possibly rather ideally, the appearance of pyrite undergoing alteration, have already been referred to in contrast with the forms of primary pyrite, as shown in Figs. 1 to 3. When the process of enrichment has gone almost to completion, there may remain only widely-separated dots of pyrite, arranged in orderly rows corresponding to the directions of the original fracture systems, as indicated in the central portion of Fig. 6.

As stated above, chalcocite is the usual derivative of pyrite and appears to be the normal product when pyrite is attacked by the acid cupriferous solutions that are the characteristic agents of enrichment. Under certain conditions that appear to be unusual and as yet seem only to occur in deepest-seated portions of the zone of enrichment, the effect upon pyrite may be different: instead of jumping from one chemical extreme, pyrite, to the other extreme, chalcocite, with no intervening

⁹ Under this heading will be considered in this paper only those types of alteration which result in the formation of other sulphides (native copper also included). The phases of decomposition which yield the oxidized products like limonite, malachite, etc., are touched upon little, if at all.

steps, there is sometimes formed around the pyrite a narrow border of secondary chalcopyrite; this in turn may be surrounded by bornite, which constitutes thus a tertiary step; then may come quaternary covellite, and finally chalcocite. In no single case have all these five minerals been found in such concentric zones, but they have all been found within the same specimen, and four of them, with either covellite or chalcocite omitted, have been found together in several cases. Further reference to this subject will be made in the description of the respective minerals involved. The cause of this plainly unusual type of alteration is not clear; the only explanation that has suggested itself being the possibility that in some occurrences the downward-penetrating solutions, while still containing copper at the lower limits of the zone of enrichment, may have had their supply of acid depleted by reactions higher up, and that under these conditions secondary chalcopyrite and secondary bornite could be formed and could maintain a precarious equilibrium for a brief period. Whether or not this hypothesis accords with the chemistry of the case is not known, but this knowledge should soon be gained, as a comprehensive chemical investigation of the copper sulphide minerals has already been begun by the Geophysical Laboratory of the Carnegie Institution, under the direction of Dr. E. T. Allen. Chalcopyrite occasionally develops in narrow vertical fractures directly under completely oxidized gossan, here, too, possibly under conditions of unusually deficient acid-supply, but the chalcopyrite so formed passes quickly to covellite.

Secondary Derivation.—Our studies have not yet extended to those cases in which true solfataric action is supposed to have taken place. In the material we have examined, however, pyrite is never secondary in the sense that it is the direct and visible alteration-product of another sulphur- or iron-bearing mineral. Where found in other than its characteristic fashion, described above as primary, it plainly occurs as later coatings, as on enargite from Morococha, Peru, or in later veins that cut all the other minerals of the ores; and so far as can be seen, it has been introduced wholly and never derived in any part from surrounding materials. While the foregoing statements appear justified, it must be admitted that in the few cases where such

later veinlets have been seen, as for example in the Wild Cat mine, Santa Rita, N. M., their significance is not clear.

2. *Pyrrhotite.*

Primary Character.—Like pyrite, the iron sulphide, pyrrhotite, merits consideration with the copper minerals because of its common presence in copper ores, especially those of eastern North America. Its occurrence in the important deposits at Sudbury, Ontario, and in other nickeliferous ores has been well described by Campbell and Knight,¹⁰ who show that in all the cases they studied the mineral is older than the associated chalcopyrite. This conclusion, as well as their description of the habit and properties of the mineral, corresponds in the main with our findings in pyrrhotitic copper ores from Georgia, Massachusetts, and Vermont, but in the important deposits of Ducktown, chalcopyrite and, to some extent, sphalerite, are intimately intergrown with some of the pyrrhotite, producing in certain instances almost micrographic structure. Even in this district, however, the greater portion of the chalcopyrite appears to be later than the pyrrhotite. In several specimens has been noted a tendency of this mineral to form wavy, vermiform stringers, surrounded by minerals of later crystallization, Fig. 9.

Alteration.—From the scanty evidence seen, we conclude that pyrrhotite undergoes enrichment in a similar way to pyrite, but that chalcocite, the sole product observed, forms somewhat more readily from the brownish sulphide than from the yellow one. Further study of the alteration of this mineral is especially desirable.

Secondary Derivation.—No indications have been found that pyrrhotite may be derived from other sulphides. Probably it behaves like pyrite in this respect, but the cases examined are not sufficiently numerous to yield safe deductions.

3. *Chalcopyrite.*

Primary Character.—Chalcopyrite has a very wide range of occurrence and closely follows pyrite as the most common sulphide mineral in copper ores, though not necessarily the most plentiful. Though primary in the majority of cases, it is always

¹⁰ Campbell, William, and Knight, C. W., On the Microstructure of Nickeliferous Pyrrhotites, *Economic Geology*, vol. ii, No. 4, pp. 350-366 (June, 1907).

of a later period of crystallization than pyrite and usually contemporaneous with or later than the gangue. On this account it rarely exists in crystals of any perfection, but rather in masses and patches of irregular outline, which, in many cases, may be shown by etching to consist of an aggregate of individual grains.

Its commonest occurrence is in highly pyritic ore as shreds and patches between and around the grains of pyrite, and as blebs or globules included within them. This type of occurrence is characteristic of the primary low-grade ores of cupriferous pyrite, though other copper-bearing minerals than chalcopyrite—for example, bornite and chalcocite—may occasionally contain the copper in the ore. In any event, it appears certain that the copper of such low-grade ores exists almost or quite entirely in the form of such definite copper minerals, since polished sections of specimens assaying as low as 0.05 per cent. of copper have been found to show grains of chalcopyrite in the pyrite. This is true of very low-grade specimens from such widely-separated localities as California, Arizona, Cuba, Spain, and Russia. Fig. 1 is an illustration of a pyritic ore containing 1 per cent. of copper from the Copper Queen mine, at Bisbee. It is the final member of a series of primary pyritic ore specimens from that mine, ranging in copper content from 0.1 to 1 per cent., which was selected for the purpose of testing this matter of occurrence of copper in such ores. It was found in this suite that the amount of chalcopyrite accorded well with the assay; and it seems safe to conclude that if pyrite contains chemically-combined copper to any extent at all, the amount is negligible. More likely pure pyrite is not at all cupriferous from the chemical standpoint.

Where present in greater amount, chalcopyrite commonly forms the matrix or cement of pyrite, as explained under the head of that mineral. In such association the chalcopyrite is in many cases intergrown with bornite, as typically represented in Fig. 2. In such cases smaller amounts of primary chalcocite are sometimes present. Its commonly later formation than pyrrhotite and its intimate, almost graphic, intergrowths with that mineral in association with sphalerite at Ducktown, have already been mentioned. In some rich primary ores pyrite may be sparing or entirely wanting; in such, bornite is a common accompaniment of the chalcopyrite, and occasionally is seen an approach towards graphic structure, as in Fig. 7, which probably

represents contemporaneous solidification practically at the eutectic composition.

When associated with sphalerite, the massive grains of chalcopyrite are commonly of later formation, but in many instances the zinc-blende is filled with minute chalcopyrite particles, many of which possess triangular outline and are probably sphenoidal crystals.

Alteration.—Chalcopyrite readily undergoes alteration to other cupriferous minerals under conditions of enrichment. As in the case of other varieties, chalcocite is the most prevalent product of this change. The enriching solutions, entering along sub-microscopic fractures, are evidenced first by thin hair-like veinlets, which gradually grow and extend until finally the whole mass of the chalcopyrite may be converted into chalcocite. Figs. 10 to 13, inclusive, show progressive stages of this alteration. In addition to the predominant chalcocite, however, covellite is formed in numerous instances, usually as aggregates and clusters of minute needles or plates that border the edges of the growing chalcocite veinlets and that are most likely to appear at an early stage of the enrichment before the development of chalcocite has become considerable. The fuzzy groups of these needles bordering veinlets of chalcocite are shown highly magnified in Fig. 10, penetrating into and encroaching upon the chalcopyrite. In this particular specimen, which is from Nacozari, Sonora, an unusual circumstance is the formation of a very narrow central line of covellite in the chalcocite veinlet, the second generation of covellite indicating recurrence of stability conditions that had disappeared while the chalcocite was forming. Occasionally covellite alone develops from chalcopyrite without the subsequent formation of chalcocite, Figs. 3 and 14, but this is probably only a preliminary stage, and if enriching conditions continue chalcocite doubtless will result.

In some specimens chalcopyrite is surrounded by a cloudy bluish pink halo, of which an indication may be gained in Fig. 13, and when the residual kernel of chalcopyrite has just disappeared, there may remain in the chalcocite a patch of this material, which, however, readily changes to chalcocite. When examined with high power this material is found to be secondary bornite, but as a rule it is a fleeting phase of the chalcopyrite alteration and evidently must possess an exceedingly narrow

range of stability in the presence of the solutions which accomplish enrichment. Characteristic development of bornite from chalcopyrite is shown in Fig. 21. Areas of the chalcopyrite of considerable size are breaking down to bornite throughout their whole area, indicating an almost utter absence of stability under the conditions existing. This decomposition is similar in character to that more often shown: bornite altering to chalcopyrite (compare Figs. 15, 16). It seems to be either more readily accomplished from secondary than from primary chalcopyrite, or else, more probably, the conditions favoring the formation of secondary chalcopyrite and secondary bornite are similar.

Secondary Derivation.—Not only is chalcopyrite subject to easy alteration, but it is itself a rather common secondary product of other minerals, though rarely produced in abundance. Its derivation from pyrite has been described under that material. It is a frequent result of the alteration of bornite, to be considered later; it is rarely developed in sparing amount from covellite at Butte; and there are indications that secondary chalcopyrite results from alteration of enargite of that district.

No direct means has been yet discovered for distinguishing between primary and secondary chalcopyrite, and all conclusions of this sort must rest upon the evidence furnished by structures and association. The shapes of the individual grains of chalcopyrite intergrown with bornite in Figs. 2 and 7 to 9 are absolutely typical of primary structures,¹¹ as proven in great numbers of cases where all evidence combines to make it certain that secondary changes have not taken place. On the other hand, the presence of the veinlets of secondary minerals and the shape of the resulting areas into which the chalcopyrite is divided, as shown in Figs. 10 to 13, are equally typical of secondary structures. Chalcopyrite that is itself secondary is shown between pyrite remnants and also scattered in cloudy patches in the bornite at the right hand of Fig. 6, also as developed secondarily from bornite in Figs. 15 and 16, which are figures of the same specimen at different magnifications, and as gashes along the cleavage of bornite in Figs. 17 and 18. When developed secondarily in areas of considerable size, as in Fig. 15, it is usually clouded with inclusions of the chemically similar bornite, which

¹¹ In this connection, compare the conclusions of F. C. Lincoln, *Economic Geology*, vol. vi., No. 3, p. 296 (Apr.-May, 1911).

may be either residual remnants of the main bornite grain which chalcopyrite has replaced, Fig. 16, or may be bornite derived from the chalcopyrite itself, Fig. 21. It is possible, however, that important areas of secondary chalcopyrite might develop without such bornite inclusions, and in that case, if surrounding relations did not afford evidence, it might well be impossible to determine whether the chalcopyrite were primary or secondary. This doubt applies to some of the chalcopyrite of the Butte ores, which, though a minor constituent of them, is much more common than has heretofore been supposed. Part of the chalcopyrite is undoubtedly primary; part is unquestionably secondary after bornite and rarely after covellite; but some, closely associated with enargite, may be either primary or secondary—not improbably both phases are present.

4. *Bornite.*

Primary Character.—Bornite, which has long been supposed to be one of the characteristic products of secondary enrichment, in reality proves to be primary in the majority of its occurrences, and its rather common occurrence in the enriched zone is found to be mainly as residual masses which are undergoing alteration. It is formed at about the same period of solidification as primary chalcopyrite and primary chalcocite, with which it is most commonly associated. Accordingly, it usually develops in irregular-shaped masses, frequently intergrown with these other sulphides, existing either by themselves or filling interstitial spaces between grains of gangue, Fig. 8, or pyrite, Fig. 2, and other ore minerals like sphalerite, Fig. 24.

Bornite very rarely occurs alone or with only gangue minerals in masses of more than microscopic size, though its intergrowth with the other copper sulphides is usually so fine-grained as to escape detection except when polished and highly magnified. Such association of chalcopyrite and bornite is illustrated in Fig. 7, and in Figs. 23 and 31 are shown beautiful graphic intergrowths of bornite with chalcocite. Small patches of such graphically intergrown bornite and chalcocite are intimately associated with enargite in ores from Butte. Bornite also occurs with enargite in the Butte ores in a peculiar relation, sometimes forming even border zones for the triangular enargite crystals and sometimes intergrown with such enar-

gite crystals. It occasionally shows graphic intergrowth directly with enargite(?) and with galena.

The common intimate intergrowth of primary bornite with other copper or copper-iron sulphides has been mentioned above, and the readiness with which bornite suffers a general chemical breakdown to form products likewise closely related to it in composition will be described under alteration. These two characteristics unite to prevent the common occurrence of masses of pure bornite of any appreciable size; yet the common development of the primary intergrowths and of the secondary alteration-products on a scale only microscopic disguises this fact. In consequence, much bornite that appears to the eye, or with the pocket lens, to be practically pure is found to contain one or more other minerals in amounts even up to 50 per cent. This condition is so general that it is safe to conclude that the great majority of analyses of bornite have been made upon material of various degrees of impurity; and it is not surprising, therefore, that these analyses show wide variations and that the formula for the mineral is still in doubt. Whether pure bornite is constant or variable in composition cannot be known until analyses have been made of pure material from several localities.

Bornite possesses three directions of cleavage, two or all three of which are made evident when decomposition sets in along these cleavage-lines, Figs. 17 and 18, or when the mineral is etched, Fig. 20. Etching, even with reagents of insufficient strength to develop the cleavage, produces at irregular intervals over the surface small rectangular solution-pits of which no hint is given in the unetched surface, Figs. 20 and 31. The reason for such differential solution is unknown.

Occurring in primary bornite, more rarely in chalcopyrite, from several localities have been found minute grains of a mineral of somewhat greater hardness than bornite, and in many, but not all, cases crystallized, probably as cubes. The color is pinkish yellow, but appears variable, and the yellower grains are more commonly encountered. This mineral appears comparatively resistant to enrichment, and usually remains unattacked after the bornite or chalcopyrite that originally surrounded it has been converted into chalcocite, but its alteration to chalcocite has been noted. Its color and association suggest

that it is related to bornite and chalcopyrite, and it may be cubanite, CuFe_2S_4 , or chalmersite, CuFe_2S_3 . But in its notable stability it does not resemble bornite or chalcopyrite, and it may be quite different from them in composition.

Alteration.—Bornite alters with exceeding readiness under conditions of enrichment. There appear to be two types of such alteration. One type is represented by the usual conversion to chalcocite along the boundaries of bornite grains, Fig. 24, and along fractures, Figs. 21, 25, 26. With continuation of this process the bornite is gradually reduced until only small remnants are left, Figs. 33, 34, and finally disappears, leaving only massive chalcocite.

The boundary between the bornite and the invading chalcocite may be sharp and distinct, as in Figs. 21 and 26, or may be indefinite and hazy, as shown in Figs. 13 and 25, even when examined with the highest powers. In the one case chalcocite undoubtedly forms directly from bornite; in the other case, the hazy border may be due to development of sub-microscopic quantities of a transition mineral; it is barely possible that this may be covellite,¹² for there is a suspicion of blue in the color of this hazy zone, and covellite is known to form readily from bornite and then be converted into chalcocite. We are inclined to believe, however, that if this hazy zone represents a distinct mineral, it is not covellite or any other ordinary variety. In the specimen illustrated in Fig. 34, there is seen (at the left of the view) a remnant of bornite undergoing alteration and surrounded by its alteration-product, which is labeled chalcocite. A part of this chalcocite around the bornite is of slightly bluer color than the normal secondary chalcocite farther away from the bornite, and it also shows distinct cleavage-lines (inherited from the bornite, as described under chalcocite), which may be visible without chemical treatment, or simply by attack of acid fumes in the air, and which the outlying chalcocite does not show. The line of demarcation between the massive chalcocite and the bluer, cleavage-showing material is fairly distinct. We regard it as possible that this material may be the same as the common bluish hazy borders to the bornite, Fig. 25, but in this particular case, instead of a very narrow band, a considerable zone of

¹² Study of material received after the first printing of this paper adds likelihood to the idea that this material is covellite, with chalcocite.

the material has temporarily withstood alteration to chalcocite. Fig. 33 appears to show the same thing more plentifully but less distinctly; in this specimen likewise the inherited bornite cleavage has been rendered visible by only atmospheric (acid fume) etching, and the color is deeper than that of the unetched chalcocite, a little of which is shown in the upper right-hand corner of the photograph. There remains, of course, the possibility that the material of this hazy zone is not a third mineral, but a sub-microscopic penetration of the bornite by chalcocite — a skirmish-line thrown out in advance of the main front of the chalcocite invasion. Such an explanation might account for the color of this hazy zone as the blended result of mixtures of the two minerals, but it seems hardly adequate to account for the greater susceptibility to chemical attack shown by the possibly similar material in Figs. 33 and 34.

This indefinite border zone between bornite and the chalcocite to which it is altering forms a possible connecting-link with the second type of bornite alteration, which is perhaps more common than the one already outlined, and which itself has a rather wide range of character. Under certain conditions when acid cupriferous solutions gain access to bornite, equilibrium appears to forsake it and it decomposes with extreme readiness. In the case of other minerals which are influenced by enrichment, such as pyrite, chalcopyrite, and covellite, it appears generally necessary to bring the altering solutions into actual contact with the portion affected before any change takes place; but with bornite it appears that as soon as the solutions reach one portion of a grain the whole area may become unstable and break down so that instead of finding the usual growing veinlet of the secondary mineral traversing and gradually replacing the one undergoing alteration, we often find in the case of bornite indefinite areas over part or all of which minute particles of the secondary minerals have developed. Most commonly this tendency is illustrated by the occurrence of an irregular zone, between unaltered bornite and final chalcocite, in which are formed the typical fuzzy aggregates of secondary covellite nearest the chalcocite, with blades and gashes of secondary chalcopyrite extending away from them into the bornite. These zones or areas of alteration may increase until they are of considerable extent and may seem to bear no

relation to fractures or other channels for the entrance of the altering solutions. Careful examination, however, usually reveals the fact that these secondary minerals occur in lines, or more commonly in gash-like forms, which represent the cleavage-planes of the bornite, and it is probably in many cases due to this cleavage that alteration can take place almost simultaneously over a whole grain, Figs. 15 to 18. A common feature in bornite thus nearly decomposed is the straight, narrow strips which have resisted the general attack, Figs. 15, 16. The reason for this is not clearly known, but it seems likely they represent portions bounded by adjacent planes of one direction of cleavage and in which the other two sets of planes do not extend or are but poorly developed.

In not all cases does the cleavage seem to account for this general decomposition of the mineral, Fig. 19, and the conclusion is inevitable that under certain conditions bornite undergoes practically spontaneous or automatic decomposition, probably with little addition or subtraction of elements, for the resulting covellite and chalcocite, richer in copper, are balanced by the chalcopyrite, which is lower in copper and higher in iron. The similar but much less common automatic alteration of chalcopyrite is shown in Fig. 21. The order of formation of the alteration-products in this type of bornite decomposition appears most commonly to be: first, chalcopyrite in very small amount and unstable; second, covellite in greater but still small amount, a little more stable; third, chalcocite, the final and stable product. In a few cases the order is reversed, so that chalcopyrite forms the central portion of gashes with covellite or chalcocite, or both, between it and the bornite, or chalcocite may surround covellite, Fig. 18. Occasionally also the relative proportion of the alteration-products differs from that stated, sometimes chalcopyrite, Figs. 15 to 17, sometimes covellite, Figs. 18, 19, being most abundant. In many of the most instructive occurrences the alternating particles of chalcopyrite, covellite, chalcocite, and residual bornite are so fine-grained and intimately disposed that their resolution into individual components is possible only with the highest powers, and photographic reproduction in black and white is well-nigh impossible. This great instability of bornite and the complexity of its alteration are

among the most interesting and beautiful features we have met with in all this work.

Secondary Derivation.—Instead of being a characteristic product in the zone of enrichment, as has generally been assumed, bornite is found to occur there for the most part only as a residual primary mineral and, indeed, as shown in the foregoing section, to be ordinarily most unstable in the presence of conditions that produce enrichment. Under certain peculiar circumstances, however, secondary bornite is formed, as mentioned under pyrite and chalcopyrite and as shown in Figs. 5, 6, 13, and 21. Its formation in such cases is best shown in Fig. 21, where it is seen gradually penetrating and replacing secondary chalcopyrite which, in turn, has been derived from primary pyrite, of which a few minute remnants remain. In most cases, indeed, it is developed only from chalcopyrite, but at Butte it is sparingly produced by the alteration of primary covellite, and in a few cases it replaces enargite, though in relations that are complex and not yet understood. Though identified at Bisbee, Morenci, Patagonia, and Superior, in Arizona; Estey, N. M.; and Butte, Mont., it is not characteristic of the ores of these districts, and even in the specimens that contain it is a subordinate constituent. Reference has already been made to the fact that this development of secondary bornite seems confined to the lower portions of the zone of enrichment; and the possibility has been mentioned that deficiency of acid in the cupriferous solutions may have permitted this mineral, and chalcopyrite as well, to form and to exist temporarily. Whatever the causes, the equilibrium between these two copper-iron sulphides appears under certain circumstances to be most delicate and liable to swing in either direction under the slightest influence.

5. *Covellite.*

Primary Character.—At least in occurrences that may be regarded as ore deposits, covellite has been generally looked upon as a secondary mineral exclusively. Our studies have been able to reverse this view for at least one locality, where a primary origin seems certain.

In the deeper levels of some of the Butte mines, covellite is found in groups of thin plates that measure up to an inch or more across, which appear in some cases to be wholly unat-

tacked by the secondary minerals, such as chalcocite and chalcopyrite, developing around them, though in other instances the covellite is being altered to secondary chalcopyrite, chalcocite, or even in small amount to bornite. Large blades of covellite have in many places been bent and broken, and the angular spaces between the resulting fragments and the lenticular gashes, formed by buckling and opening up of adjacent plates, have been filled with later, but nevertheless primary, bornite which is now being converted into chalcocite, Fig. 34; but the covellite itself shows practically no sign of attack and retains sharp, angular boundaries. The appearance certainly is strongly indicative of the free crystallization and subsequent deformation of covellite before any other mineral was deposited. In other specimens long, exceedingly thin microscopic blades of covellite are included within rounded or irregular grains of pyrite of otherwise typical character, which are themselves embedded in a solid ground-mass of larger covellite plates; the structure of the covellite inclusions in the pyrite is identical with the ophitic type of poikilitic penetration of pyroxene by feldspar laths in diabase. It is impossible to believe that these extremely slender plates of covellite, undistorted and of typical habit, can have forced their way by alteration into pyrite grains which otherwise are perfectly fresh, and there is every reason to believe that the pyrite is as primary here as it ever is. The only conclusion, therefore, seems to be that the covellite is primary and that a little of it started to crystallize before the pyrite, but that most of it crystallized after the pyrite had formed around the earliest blades and, shutting them off from further accessions of covellite-forming material, had preserved them in the thin delicate blades now seen. Unfortunately this occurrence was discovered too late to permit reproduction here of a photograph of it, which it is believed would be entirely convincing.

In material from Shasta county, Cal., areas of bornite, sphalerite, galena, tetrahedrite, and gangue in typically primary association are cut across by sharply-defined, broad plates of covellite. The relations here also suggest that the covellite is primary, but the case is not so clear as in the material described above.

In distinction to the rather dull luster, generally deep blue

color, and always feathery or aggregate structure of covellite which plainly has formed secondarily, these larger masses of possibly primary covellite are bright and massive, but the color ranges¹³ from fairly light to deep blue and may show a considerable variation even within a single individual.

Alteration.—Covellite yields easily to the attack of decomposing solutions, and its conditions of stability appear to have a rather narrow range. The product of such alteration appears almost invariably to be chalcocite, but occasionally chalcopyrite or a small amount of bornite is formed from it.

Secondary Derivation.—Covellite is frequently produced from other sulphides, and if it were itself more stable, it would be a rather common mineral of the enrichment zone. It develops most commonly from bornite, Figs. 18, 19, as has been described in some detail under the heading of that mineral; but, except in rare instances, it is the product of a transitory period of such alteration, and therefore is not formed plentifully, being quickly replaced by chalcocite, as a rule. It readily forms also from chalcopyrite, Figs. 3 and 14; but here, likewise, usually gives way to chalcocite after the early stages of enrichment. The development of covellite from chalcocite, indicating a reversal of the usual order, is occasionally seen, Fig. 10. Its rather common formation as veinlets in sphalerite, Fig. 35, may be significant as indicating a direct and equal interchange of metals, in distinction to the more complicated reactions which probably are generally involved in the formation of the secondary sulphides.

The characteristic blades of secondary covellite, which may coalesce into aggregates of feathery appearance, have been already referred to.

6. *Chalcocite.*

Primary Character.—Chalcocite furnishes, at the present time, considerably more than half of the world's supply of copper. The mineral has long been known as the most important sulphide occurring at shallow depths, and since the announcement of the theory of secondary sulphide enrichment, chalcocite has quite generally been regarded as a universally

¹³ For significance of color change, compare M. Lazarevic, *Zeitschrift für praktische Geologie*, vol. xx., No. 9, p. 355 (Sept., 1912).

secondary mineral. After referring to certain occurrences in which the mineral had been described as primary, Dr. F. L. Ransome recently summarized the general opinion as follows: "The burden of proof appears to rest at present upon those who support the view that chalcocite can be deposited simultaneously with quartz and pyrite or other sulphides characteristic of unmodified or primary veins."¹⁴ Shortly afterwards Dr. F. B. Laney¹⁵ published results of study by the metallographic microscope, in which he showed, beyond all reasonable doubt, that chalcocite is primary. He appears to have been the first to picture and describe the beautiful micrographic intergrowths of bornite and chalcocite¹⁶ which prove contemporaneity and which we have now found formed of these and other minerals from various regions. As a matter of fact, our studies indicate that primary chalcocite is comparatively common, but that in most places it is present in but small amount. At Virgilina, Va.-N. C.; Butte, Mont.; Tintic, Utah; Tularosa, N. M.; Bisbee, Ariz.; Cananea, Mexico; and Bonanza mine, Alaska, however, are deposits in which important amounts of apparently primary chalcocite occur; and in the cross-veinlets encountered in certain of the Lake Superior copper mines the chalcocite also appears to be primary. The mineral is lacking in crystalline outline, but in the primary occurrences develops a granular crystalline structure, which is evidenced by the fairly perfect cleavage that the mineral possesses; a fracture-surface of such massive primary chalcocite therefore exhibits a faceted appearance, similar to that of marble. The polished surface of primary chalcocite varies somewhat in color from white or bluish white, which is most common, to darker shades distinctly bluish or, in some cases, pale pinkish or greenish. In general, however, primary chalcocite is of lighter color than the secondary phases of the mineral, which are characteristically more bluish, but the color-difference is not sufficient to permit safe discrimination on this ground alone.

¹⁴ Criteria of Downward Sulphide Enrichment, *Economic Geology*, vol. v., No. 3, p. 211 (Apr., 1910).

¹⁵ The Relation of Bornite and Chalcocite in the Copper Ores of the Virgilina District of North Carolina and Virginia, *Economic Geology*, vol. vi., No. 4, pp. 399-411 (June, 1911).

¹⁶ The specimen illustrated in Fig. 31, showing such a micrographic intergrowth, was kindly given us by Doctor Laney, and is taken from the same occurrence in which he found these structures.

In some occurrences large masses of ore contain no other mineral than chalcocite. More commonly, however, it is intimately intergrown with other minerals, most frequently with bornite, but also with enargite and less commonly with chalcopyrite. With bornite, it not uncommonly exhibits micrographic structure, Figs. 23 and 31, especially in portions of an ore where other minerals are not nearby. When such a specimen has been etched to bring out the cleavage of both the bornite and the chalcocite, it is found that the various portions of each mineral, separated by similar masses of the other, possess cleavage all in the same direction; in other words, that these intergrowths are of skeleton crystals of the two minerals. In some cases a structure of intergrown bornite and chalcocite, Fig. 22, which may be called sub-graphic, corresponds to a structure of bornite and chalcopyrite, Fig. 7. The commonest type of intergrowth of chalcocite with bornite and more rarely chalcopyrite is less perfect, crystallographically, and is wholly similar to the intergrowth of bornite and chalcopyrite shown, for example, in Fig. 2.

Chalcocite, which from broad field-relations and microscopic character appears to be primary, is found to develop in all cases a remarkably good cleavage when etched with acid. Fig. 27 illustrates an etched surface of typical ore from the great chalcocite mass of the Bonanza mine, Alaska, and Fig. 28 shows natural etching developed by weathering in a piece of chalcocite ore from the dump of the old Schuyler mine, N. J. The cleavage in these may be compared with that shown in Fig. 29, which is an etched face of synthetic chalcocite produced by fusion of copper and sulphur at the Geophysical Laboratory, Washington, D. C., and generously given to us for study by Dr. E. T. Allen. In reality, the mineral possesses three cleavages, at right angles, though of course the chance direction in which the polished face cuts these cleavage-planes may give varied cleavage-patterns.

Alteration.—Chalcocite, the mineral richest in copper of all the sulphides, constitutes the normal end member of the series of minerals developed by secondary sulphide enrichment, and may be regarded as the stable product towards which the process tends. Accordingly, primary chalcocite, occurring in an ore subjected to enrichment, commonly shows no alteration,

and the veinlet of secondary chalcocite that may come up to a grain of primary chalcocite from both directions stops at the boundary-line, even though the central fracture, from which the secondary veinlet grew, may continue right through; see bottom of Fig. 23. In certain cases, however, chalcocite is converted into other minerals. An illustration of this is shown in Fig. 10, where a veinlet of chalcocite has been converted along its center into covellite. Covellite and chalcopyrite also form sometimes at the expense of chalcocite, where all three are being derived from the decomposition of bornite; but apparently in all such cases only small quantities of these other minerals form at the expense of chalcocite, and in a general way they may be regarded as relatively unstable as compared with it.

Since chalcocite is the prevalent mineral of the secondary sulphide zone and therefore reaches up to the horizon of active oxidation, it is the one of the secondary sulphides most likely to suffer oxidation. Its conversion into the oxidized minerals, such as cuprite and the carbonates, we have not studied in detail, but we find that under conditions of direct oxidation the molecule of sulphur which the mineral carries is oxidized away, leaving metallic copper in intimate association with the chalcocite, Fig. 32. Veinlets of metallic copper of similar appearance, but containing also native silver, are found in cross veins of chalcocite in the Lake Superior copper region, but it is not yet established that this latter occurrence is a result of oxidation. Conditions favorable for the development of native copper from chalcocite appear not often to exist, as this product is not common nor, in most of the cases where it does exist, abundant. More frequently the chalcocite passes, by more complete oxidation, to the cuprous oxide, cuprite, and from this by the action of CO_2 to the carbonates. In considerable part, however, the chalcocite appears to be oxidized at once to a soluble condition, probably by direct addition of oxygen to form sulphate. The result of this action is the one most commonly seen under the microscope. The product is a dull, more or less incoherent and soft residue which is commonly described as "sooty" chalcocite. As has been made evident in the field-examination of numerous occurrences and thus confirmed by the microscope, this sooty chalcocite represents a stage of incip-

ient decomposition, instead of indicating, as some have thought, an early stage of deposition of chalcocite, which, in reality, is indicated by a bright, lustrous appearance of the mineral. This solution and attendant disintegration of chalcocite begins along minute fractures, which are emphasized especially when the polished surface is etched, Fig. 12. Most narrow secondary chalcocite veinlets show a narrow dark line along their center, Figs. 21, 25, 26; this doubtless represents the original fracture from which the veinlet has expanded, and its dark appearance indicates that a depression below the general surface exists there, probably caused by tearing out during polishing of the disintegrated sooty chalcocite along this channel.

Secondary Derivation.—Chalcocite is the commonest of the secondary copper sulphides because it is the one possessing the greatest stability under enriching conditions. Mention has been made, in the description of the preceding minerals, of their attack and replacement by chalcocite. The most common example of this is afforded by the conversion of pyrite, Figs. 4, 5, 6, 21. The alteration described is characteristic both of heavy pyritic ores and of disseminated pyrite grains in the rocks. It is the enrichment by partial or complete replacement by chalcocite of such separated poorly-cupriferous pyritic grains in a rock that furnishes the typical ore of the great porphyry copper deposits, such as those of Bingham, Morenci, and Miami. Reference has been made to the fact that under certain unusual circumstances chalcopyrite, bornite, and covellite may appear as products of an intermediate stage of decomposition of the pyrite into the final chalcocite.

From chalcopyrite secondary chalcocite develops, as has been mentioned in connection with Figs. 10 to 13, with bornite as an intermediate stage in the last instance. Chalcocite also develops directly from bornite, Figs. 25 and 26, as described under the alteration of that mineral. Many of the micrographic intergrowths of primary chalcocite and primary bornite at Butte have slightly hazy boundaries between the two minerals, evidently due to the very beginnings of conversion of the bornite into secondary chalcocite. More frequently in the alteration of bornite, chalcocite is formed after some covellite and a little chalcopyrite have been developed as an intermediate stage, Figs. 18 and 19. The typical feathery and fuzzy secondary covellite formed from chalco-

pyrite or bornite alters readily to chalcocite, but the compact masses of primary covellite like that in Fig. 34 appear surprisingly resistant, even though in the former case secondary chalcocite is developing all around the covellite. From pyrrhotite the formation of chalcocite resembles that from pyrite, except that it seems to take place a little more easily and, so far as observed, without the formation of intermediate products. From enargite chalcocite forms with difficulty, so that residual grains of this arsenical mineral persist in the chalcocite for a long time, but finally are replaced. Our data are meager regarding the alteration of tetrahedrite and tennantite. These minerals seem to be even more resistant than enargite to chalcocite replacement, but tennantite at least finally suffers this change, as shown in specimens from Butte and from Tanalyk, Siberia. From bivalent sulphides, sphalerite and galena, chalcocite forms rarely, and usually then only as narrow veinlets, Fig. 24, across these minerals which more commonly alter to covellite,¹⁷ Fig. 35.

The cleavage characteristic of primary chalcocite and its development by etching have already been described. When chalcocite that is plainly secondary is etched, it usually shows, instead of cleavage, countless irregular cracks. (See Fig. 30. The specimen shown in Fig. 12 was etched and showed similar cracking, but it has since been repolished so only the deepest cracks remain.) These closely-spaced cracks commonly form the outlines of individual grains, within which imperfect cleavage can sometimes be discerned, but their boundary-lines etch more deeply than such cleavage-planes, and are therefore more noticeable. In general, such secondary material, when etched, can be distinguished without difficulty by its irregular cracking and small grain from the large individuals with good cleavage which appear when primary chalcocite is etched.

The reason for this difference in perfection of cleavage exhibited by primary and by secondary chalcocite is not perfectly clear, but it may possibly be due to the fact that primary chalcocite probably crystallizes from solution only because the conditions for its separation are favorable, and thus a fairly perfect and self-characteristic molecular structure would result, at least internally, and would find expression in good cleavage, wholly

¹⁷ See foot-note No. 21.

and solely characteristic of chalcocite; but when secondary chalcocite is formed, by gradual replacement of some existing sulphide of different molecular structure, the chalcocite is obliged to overcome this antagonistic structure before it can assume its own characteristic arrangement. This evidently it ordinarily fails to do with any completeness, therefore with the structure of the original sulphide partly imposed on the secondary chalcocite, the resulting molecular arrangement is imperfect and confused, under which circumstance good chalcocite cleavage cannot be realized. The extreme of such superimposition of structure is shown in the bodily inheritance of bornite cleavage by secondary chalcocite, as described in the following paragraph. Another prominent factor against the development of large secondary chalcocite individuals with good cleavage is the fact that instead of growing from a single focus, as most primary grains do, masses of secondary chalcocite form by the coalescing of many originally separated and wholly independent small areas, each with its own crystallographic orientation; the resulting final mass is thus composite, and any cleavage that may exist in it extends over only small portions of it.

Chalcocite that has developed at the expense of bornite sometimes shows a cleavage, as already mentioned under that mineral. This cleavage may be fairly perfect and visible without etching, Figs. 33 and 34 (see under bornite for significance of this). Such cleavage in undoubtedly secondary chalcocite may easily be confused with the cleavage characteristic of primary chalcocite, and indeed cannot in all cases be discriminated positively. In the case of the chalcocite secondary from bornite, however, examination shows that this cleavage is not native to the chalcocite, but is in reality an expression of the cleavage of bornite which has been retained pseudomorphously, or, in other words, inherited by secondary chalcocite. These cleavage-lines, as shown in Figs. 33 and 34, penetrate from the chalcocite into the bornite, and when such specimens are etched so as to develop the cleavage of bornite also, the lines are found to be absolutely continuous across both minerals. This is a most beautiful and delicate example of pseudomorphous replacement, but it materially reduces the reliability and value of an otherwise apparently dependable means of distinguishing primary from secondary chalcocite: viz., by the presence or ab-

sence of good cleavage. This test, therefore, while undoubtedly of importance, must be applied with caution and may even then conceivably lead in some cases to erroneous conclusions.

7. *Enargite.*

Primary Character.—As a copper ore mineral, enargite is important in but few districts. Inasmuch as most of our material containing it is from the mines of Butte, our conclusions are perhaps not well founded, but they seem to be supported by the few specimens we have studied from other localities. The mineral appears to be practically in all cases primary. It is of later formation than pyrite, but in considerable part is older than most of the other sulphides, and therefore forms well-outlined figures, sometimes of rectangular but more commonly of triangular shape. Not infrequently, as at Morococha, Peru, it occurs in radiating prismatic blades; and one specimen from that district shows enargite coated with a later envelope of pyrite and tennantite(?). In the Butte ores there is, in addition to the well-crystallized enargite, much that occurs in irregular areas intergrown with bornite, chalcocite, and sphalerite in the structure characteristic of primary and practically simultaneous deposition. In the polished section, the cleavage of the mineral is rarely shown.

Enargite crystals commonly exhibit a somewhat mottled appearance, due to a peculiar intimate intergrowth of materials slightly different in color and hardness. One of these materials is bright white; the other slightly darker and harder, ranging from lightest gray to faint brownish or pinkish. The difference in color of the two constituents is, however, often so slight that it might not be detected except where areas of the two are adjacent. In some crystals the arrangement of these differently colored areas is in alternating stripes, suggesting the twinning of plagioclase, though never very perfectly developed; in other cases it is patchy, more like the microperthite structure, but always it is intimate and consists of small areas of one alternating with similar masses of the other. Whether this phenomenon indicates that there are two forms of enargite, or that the composition of the mineral is not constant, or that it is isomorphous with some other mineral of closely related composition, we have not yet been able to determine;

but this feature is characteristic of practically all the crystalline enargite we have seen, no matter from what locality. In the case of that portion of the enargite at Butte which forms in irregular grains contemporaneous with the other sulphides, there is no such intergrowth, and this corresponds to the whiter of the two materials in the crystals.

While there are, in one sense, two generations of enargite at Butte, as evidenced by the existence of good crystals and of irregular interstitial areas, there is no evidence furnished by either the enargite or any other of the Butte sulphides to indicate that there were two distinct periods of ascending or primary mineralization, as some have thought.¹⁸ The two phases of enargite, while undoubtedly differing slightly in time of crystallization, belong as much to one period of solidification as do the magnetite and quartz of a granite, or the phenocrysts and ground-mass feldspars of a porphyry.

At Butte many of the enargite crystals contain bornite in irregularly lobed and rounded forms, probably primary inclusions, though the exact relations are not certain.

Alteration.—Enargite appears to be comparatively resistant to change by acid copper-bearing solutions, and residual crystals of it, showing little if any sign of attack, are found surrounded by secondary chalcocite. It does, however, alter in some cases, as shown by certain Butte specimens in which veinlets of covellite, or covellite with some chalcopyrite, are cutting it; and in numerous instances it contains patchy areas of chalcopyrite of suspicious appearance, Fig. 17, which probably have been derived from the enargite, though the exact nature of the solutions which accomplished this is unknown and not unlikely differed somewhat from the typical solutions usually causing secondary enrichment.¹⁹ Finally, of course, it is converted into chalcocite. Much further study can profitably be given to this feature of enargite occurrence.

Secondary Derivation.—In only one case we have found secondary enargite. This is a specimen from Butte, and the fact that some 40 other enargite-bearing specimens from that

¹⁸ Study of material just received indicates that this statement may possibly be open to question, but probably not in its application to enargite.

¹⁹ Since the above was written, crystals of enargite partly converted to bornite and chalcopyrite have been received from Mr. Sales, whose determinations we confirmed microscopically.

district fail to show it may serve to indicate how uncommon such an occurrence is. In this particular instance the specimen contains a large percentage of enargite, mainly in good-sized crystals exhibiting the two usual intergrown constituents. The spaces between these crystals are filled with irregular intergrowths of enargite, bornite, and chalcopyrite, some of which last seems to be primary but much may be secondary, derived from the interstitial enargite. A peculiar phenomenon in the chalcopyrite is the presence in it of cloudy bands parallel to but at some distance away from the faces of enargite crystals. These bands appear to be due to extremely minute inclusions in the chalcopyrite, and under highest powers they seem most like bornite, but their certain determination has not yet been accomplished. Cutting this specimen are narrow fractures along which, close to the primary enargite, a secondary generation of the same mineral has formed as veinlets that replace bornite and chalcopyrite; but this secondary enargite never extends along the fractures far from areas of the primary mineral, and apparently this exception to the general rule of primary occurrence of enargite is due to the abundance of the mineral in this particular specimen and to the possibility of solution and reprecipitation of small quantities of it when the distance of transportation is very small.

The evidence from which Lazarevic²⁰ concludes that secondary enargite is abundant at Bor in Servia does not seem especially convincing. His descriptions of the mineral from Butte make it plain that his material was not representative of typical occurrence in that district.

8. *Tetrahedrite and Tennantite.*

These two complex and closely-related sulphides have been found in a number of specimens we have examined, but our knowledge concerning them is much less complete and reliable than of the other minerals described. The evidence at hand is to the effect that they are always primary. They appear very similar on the polished surface, and usually can be distinguished positively only by chemical means. In general, they lack crystal form, but occur in irregular grains and patches, as shown

²⁰ Lazarevic, M., *Zeitschrift für praktische Geologie*, vol. xix., p. 322 (Sept., 1911), and vol. xx., No. 9, pp. 354 and 357 (Sept., 1912).

in occurrences of tetrahedrite from Cananea, Mexico, Fig. 20, and of tennantite from Tanalyk, Siberia, Fig. 36, and from Butte, Mont. At Mogollon, N. M., bands of tetrahedrite alternate with bornite and chalcopyrite.

These minerals seem to be comparatively stable and usually are not affected in the first stages of enrichment. After long attack, however, they yield, chalcocite appearing to be the most common result, but some Butte tennantite has been observed altering to chalcopyrite.

No secondary tetrahedrite or tennantite has been seen in any of the specimens examined.

III. CONCLUSIONS.

The main object of our work up to the present time having been acquisition of the facts regarding the sulphide ores of copper, deliberate attempt at systematic analysis and interpretation of these facts has been withheld till such time as the accumulation of data should be so concordant and illuminating as to indicate definitely the conclusions that might be drawn and give adequate confirmation to them. That time has not yet fully arrived; in certain directions, however, the facts already gained point indubitably, and for those matters upon which evidence is divided it has been felt that presentation of tentative deductions might be justified if it served no further purpose than to call attention to the problems and as a target for comment and criticism.

Some of the deductions that may be drawn from this microscopic study of the copper ores are therefore set forth below. To what extent these conclusions are supported and how far and how safely they may be applied as generalizations, it is hoped that each reader may decide for himself after perusing the data on which they rest, given in the preceding pages. So far as conclusions regarding the minerals are concerned, it is evident that their reliability is proportional to the frequency with which the several varieties are encountered.

Conclusion as to whether an ore or a given mineral in an ore is primary or secondary can be safely drawn in the great majority of cases from the type of structure involved. For instance, compare the characteristically primary structure in

Figs. 1, 2, 7, 8, 9, 20, 22, 23, 24, 31, and 36, with the typically secondary structures in Figs. 4, 5, 6, 10 to 19, inclusive, 21, 25, 26, 35. The significance of these contrasting types of structures could be gained originally, of course, only by knowledge of and comparison with the facts of field-occurrence. Now that these types of structure are standardized, they may be applied to unknown occurrences. Structures and relationships of doubtful significance may be encountered occasionally.

In the case of chalcocite, internal or molecular structure as expressed in cleavage may also be used with caution as a criterion of the primary or secondary character of the mineral.

The following sulphide minerals common in copper ores are primary, *i. e.*, they are products of original precipitation, probably from ascending, heated, alkaline solutions: pyrite, pyrrhotite, chalcopyrite, bornite, covellite, chalcocite, enargite, tetrahedrite, tennantite, sphalerite, and galena.

Of these, the following, so far as observed, are invariably primary: pyrite, pyrrhotite, tetrahedrite, tennantite, sphalerite, and galena.

The following minerals are secondary, *i. e.*, they are derived by the action of descending, cold, acid waters upon the other sulphides, *i. e.*, secondary sulphide enrichment: chalcopyrite, bornite, covellite, chalcocite, and enargite.

None of these minerals is invariably secondary.

In no case has a secondary sulphide been observed developing except by replacement of another sulphide mineral.

Resistance to enrichment among the copper-bearing sulphides decreases in the following order: chalcocite, enargite, tetrahedrite and tennantite, covellite, chalcopyrite, bornite.

Of the sulphides developed secondarily, chalcocite is by far the most common and abundant; covellite is common, but not usually present in large amount; chalcopyrite is not uncommon, but rarely abundant; bornite is rare and generally sparing in amount; and enargite is very uncommon and in insignificant quantity.

Stability of the secondary sulphides in the presence of acid cupriferous solutions therefore decreases in order as follows: chalcocite, covellite, chalcopyrite, bornite; enargite has the anomalous character of being rather resistant to alteration, but difficultly regenerated. The instability of bornite, as regards both ease of decomposition and difficulty of secondary formation, is especially noteworthy.

Under the usual conditions of enrichment, secondary chalcopyrite forms but sparingly, and secondary bornite little if at all. Under certain conditions, however, these minerals are developed secondarily with some readiness, though they are then subject to easy decomposition. The evidence at hand indicates that the secondary formation of these two minerals is most favored in the farthest or deepest portions of the zone of enrichment; and it is possible that deficiency of acid (or perhaps excess of iron) in the cupriferous solution is the cause of such secondary production.

Including the reactions in these deep-seated portions of the secondary zone, it is possible to pass from pyrite, the sulphide richest in iron, to chalcocite, the sulphide richest in copper, through the intermediate formation of a series of minerals which, in the order most commonly seen, represents an orderly progression of iron subtraction and copper addition. Except in the case of bornite, there is also a gradual decrease in sulphur content. Since chalcocite is the common end-product of enrichment, these conclusions as to increase of copper and removal of iron and sulphur hold good whether or not all or any of the intermediate products are formed.

This subtraction of sulphur, which can be accomplished only by oxidation (this probably applies to iron also), indicates that

in its broadest sense the process of secondary enrichment is an oxidizing process; or, stated otherwise, that it is accomplished by the reducing power of the primary sulphides. Reactions like the development of covellite from sphalerite may possibly be exceptions to this general rule.²¹

Since secondary enrichment is thus an oxidizing process, the subsequent oxidation of its final product, chalcocite, must be related to the enriching process, in its chemical tendency at least. The product of direct oxidation of chalcocite is native copper.

Arranged in order of usual formation, therefore, the minerals show the following interesting progressions:

Mineral	Formula	Per Cent.		
		Cu.	S.	Fe.
Pyrite....	FeS ₂	0.0	53.4	46.6
Chalcopyrite..	CuFeS ₂	34.5	35.0	30.5
Bornite, { old formula ..	Cu ₅ FeS ₄	55.5	28.1	16.4
{ new formula ..	Cu ₃ FeS ₄	63.3	25.5	11.2
Covellite	CuS.	66.4	33.6	0.0
Chalcocite	Cu ₂ S.	79.8	20.2	0.0
Copper.	Cu.	100.0	0.0	0.0

The elimination of the sulphur is doubtless accomplished in sulphuric acid and sulphate solutions, which are dissipated through the rocks.

One of the most puzzling matters connected with the process of secondary enrichment is explanation of how the iron, derived by enrichment of the primary ore at the bottom of the zone of enrichment, finds its way to the top of the zone to form gossan. The specimens studied afford no clue to this.

²¹ That the secondary sulphides are produced by reduction was recognized by H. V. Winchell (*Bulletin of the Geological Society of America*, vol. xiv, pp. 271, 275, 1903), and by J. F. Kemp (*Economic Geology*, vol. i., No. 1, p. 11, Oct.-Nov., 1905). Since this paper went to press the very interesting article by Dr. A. C. Spencer, has appeared (*Journal of the Washington Academy of Sciences*, vol. iii., No. 3, pp. 70-75, Feb., 1913). In this (p. 71) we are glad to find corroboration of our view that secondary enrichment is essentially a process of oxidation. The suggestion (p. 75) that covellite instead of chalcocite may commonly form from sphalerite because of favorable volumetric relations is also interesting.

Many of the reactions involved in the process of enrichment are reversible when conditions change slightly; see Fig. 10, showing development of chalcocite from covellite, and its conversion back to covellite again. This is especially true of the relations between bornite and chalcopyrite under enriching conditions, Figs. 16 and 21. Possibly the development from bornite of some chalcopyrite, signifying a backward step in the process of enrichment, is related in some way with the abnormal serial relation of the sulphur percentage of bornite, as indicated in the foregoing table.

If a low-grade primary pyritic ore like Fig. 1 is converted to chalcocite by enrichment (advanced stage of this shown in Fig. 4), it is evident that the degree of enrichment is high. If, however, the original ore already consists of rich copper minerals and then is converted to the final enrichment-product, chalcocite (as is taking place in such cases as Figs. 13 and 25), the ratio of enrichment is of course much lower. Comparison of the facts of field-occurrence with the microscopic findings indicates that Nature, like the modern jig or vanner, cannot economically produce an extremely high ratio of concentration; when attempted, loss becomes excessive and the operation fails of its object. In other words, it appears, in the case of heavy sulphide ore bodies, that a primary ore of low copper content yields on enrichment only a comparatively shallow zone of high-grade secondary ore, for when the zone becomes thicker, addition of copper to it from the gradually descending top of the primary zone is balanced or exceeded by dissipation and loss of copper during transportation in solution downward from the top of the secondary zone where oxidation and solution are at work. This conclusion appears at first thought not to apply in case of the disseminated "porphyry" ores, but in reality it does hold there also if attention be confined only to the ore minerals and the predominant rock gangue be ignored.

On the other hand, in such camps as Butte and Bisbee, where the thickness of the rich secondary zone is remarkable, it appears that, although lean pyritic ores are known to exist, the great secondary chalcocite bonanzas were derived, not from such lean ores, but from primary ores which were already rich, like Figs. 17 (slightly enriched) and 22 for Butte, and Figs. 2

and 7 for Bisbee. The enrichment in these cases was doubtless accomplished at the expense of great loss of copper; but with rich primary ores to begin with, Nature could afford to be extravagant and inefficient and still produce a high-grade product in abundance. In such deposits, therefore, it is believed that mining can be expected to continue profitably after the enriched zone may have been passed through, the difference in grade between primary and secondary ore not being extreme.

In the average primary copper ore containing pyrite, this mineral is commonly cracked and crushed, and in many instances disarranged. Whatever the cause of this may be, it is not the result of external deformation after the deposition of the ore.

It is conceivable and generally believed that a perfect crystal of a mineral can develop by replacement of other material, like garnet in schist or pyrite in shale. The graphic intergrowths, Fig. 23, now known to be rather common in sulphide ores, represent a peculiar and rather ideal type of crystallization from solution that does not seem to accord with this idea of gradual replacement, *i. e.*, piecemeal solution and coincident precipitation; rather is suggested free crystallization from solution in space. This is easily possible in such cases as the veins of filling of Virgilina, but is difficult to understand, in the light of our present conceptions of replacement, when occurring in the replacement-lodes of Butte or the contact-metamorphic replacements of Cananea. In view of the recent showing by Lindgren²² that replacement is very commonly governed by volumetric, not by stoichiometric, laws, this indication afforded by the graphic intergrowths may possibly be significant of a lapse of time in certain cases between solution, that makes a cavity, and precipitation, that fills it. If this may be true for those instances in which a graphic intergrowth results, it probably is not untrue in many cases where this particular phenomenon is absent.

Our study has revealed a remarkable uniformity in the conditions of copper ore formation. In the primary deposits

²² Lindgren, W., The Nature of Replacement, *Economic Geology*, vol. vii., No. 6, p. 534 (Sept., 1912.)

there is striking similarity of features, such as the space-relationships of the component grains, sequence of crystallization, stability to alteration, etc. Regardless of locality, structural or genetic type of deposit, or geologic age, the great majority of primary deposits represented in the specimens studied exhibit notable resemblances—whether the disseminated ores in sandstone of Atbasar, Siberia, the lodes of Butte, the impregnations of Braden, Chile, or the contact-metamorphic replacements of Bisbee and Texada Island—whether the late Tertiary replacement-lodes of Mogollon, N. M., the ancient veins of filling of Virgilina, or the great pre-Cambrian replacement-bodies of Jerome, Ariz. It is true that there are exceptions, like the chalcocite deposits of Bonanza, Alaska, and of Tularosa, N. M., but these may appear somewhat out of harmony with the rest only because there are gaps in our knowledge and conceptions.

The secondary sulphide deposits also exhibit a very noticeable uniformity of character. They vary in the degree to which enrichment has extended, and in other details of subordinate importance, but the broad fundamental features of structure, stability, sequence, and chemical significance are common to all, whether derived from a lean ore or a rich one, in a warm climate or a cold one, under humid or arid conditions. If anything, the secondary deposits are less uniform in essential features than are the primary deposits; the chemical elements involved are fewer in number in the secondary ores, but the relationships are somewhat more complicated and diverse.

In conclusion, we desire to emphasize what is perhaps the most important result of this study, viz., the abundant evidence afforded that primary copper deposits are definite, characteristic, clear-cut phenomena, of striking similarity in the main, formed by well-defined processes as a result of certain specific causes acting through a limited and probably brief period of time. Among the ores that we have studied, at least, there are absolutely no indications favoring—rather, there is plentiful testimony against—the hypothesis that many or most ore

bodies (excluding the enriched zone) are the result of "repeated reworkings" of their constituents by various processes, of different causes, at separated times or over long-extended periods. This idea, never ringing true nor supported by competent evidence, has served only to complicate and befog the situation, and thus postpone universal acceptance of the fast-growing and now almost overwhelming proof of the direct and sole influence of igneous agencies in the formation of many ore-bodies.

EXPLANATION OF MICROPHOTOGRAPHS.

In the plates on subsequent pages symbols are used to denote the various minerals, as follows :

b = bornite.	gn = galena.
cc = chalcocite.	p = pyrite.
cp = chalcopyrite.	po = pyrrhotite.
cu = copper.	sl = sphalerite.
cv = covellite.	tn = tennantite.
en = enargite.	td = tetrahedrite.
G = gangue (any transparent minerals).	

The black or very dark gray portions represent spaces or depressions below the polished surface, like fractures and polishing-pits, or they indicate transparent gangue material. Dark rims around hard minerals like pyrite, however, are due simply to oblique reflection of light where the surface of greater relief joins that of the softer mineral surrounding.

The magnification in diameters is given after the statement of locality. These illustrations represent a reduction to about two-thirds the dimensions of the original microphotographs, and the figures for magnification have been reduced accordingly. Much loss of detail and of clearness has naturally resulted from the reduction and half-tone reproduction, and in consequence certain features referred to in the text and in the accompanying descriptions are not as plain in the illustrations as could be desired.

FIG. 1.—COPPER QUEEN MINE, BISBEE,
ARIZ.
× 30.

Pyrite with granular polygonal structure and irregular interstitial grains of chalcopyrite.

FIG. 2.—COPPER QUEEN MINE, BISBEE,
ARIZ.
× 30.

Pyrite showing crushed and distorted or "exploding bomb" structure cemented by chalcopyrite and bornite in typical primary intergrowth. Small blebs of bornite included in the pyrite are indicated by arrows.

FIG. 3.—COPPER QUEEN MINE, BISBEE,
ARIZ.
× 30.

Pyrite showing typical crushed structure and somewhat distorted in a matrix of chalcopyrite which is being altered to covellite.

FIG. 4.—DETROIT MINE, MORENCI,
ARIZ.
× 36.

Strongly fractured or sheeted pyrite largely converted into secondary chalcocite.

FIG. 5.—THREE R MINE, PATAGONIA,
ARIZ.
× 12.

Shattered pyrite being altered to chalcopyrite, bornite and chalcocite.

FIG. 6.—LOWER LEFT-HAND CORNER
OF FIG. 5 AT A MAGNIFICATION OF
36 DIAMETERS.

Shows details of the alteration of pyrite into chalcopyrite, bornite, and chalcocite.



FIG. 1.

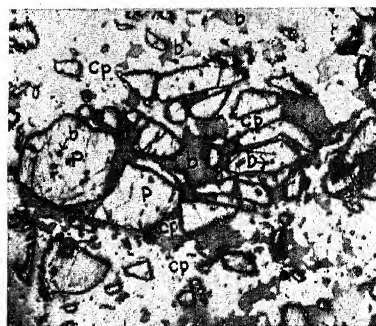


FIG. 2.

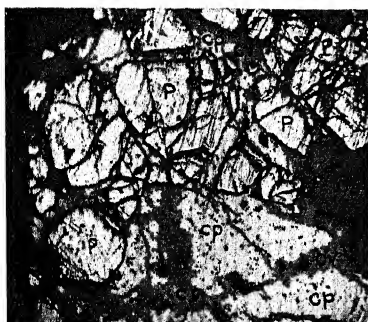


FIG. 3.

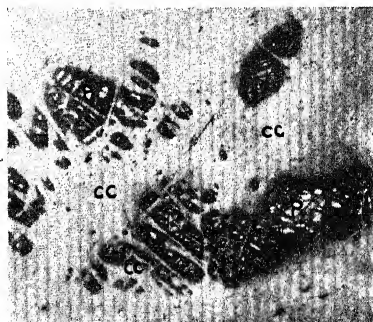


FIG. 4.

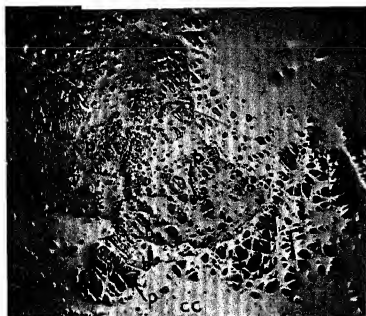


FIG. 5.



FIG. 6.

FIG. 7.—COPPER QUEEN MINE, BISBEE, ARIZ.

× 32.

Typically primary intergrowth of chalcopyrite and bornite almost graphic in character.

FIG. 8.—TEXADA ISLAND, BRITISH COLUMBIA.

× 53.

Typically primary intergrowth of bornite and chalcopyrite with some sphalerite. The gangue consists of contact-metamorphic silicates, including garnet, epidote, and needles of wollastonite.

FIG. 9.—TEXADA ISLAND, BRITISH COLUMBIA.

× 53.

Typically primary intergrowth of chalcopyrite and bornite with vermicular development of pyrrhotite.

FIG. 10.—PILARES MINE, NACOEZARI, SONORA, MEX.

× 240.

Chalcopyrite cut by composite veinlets of covellite and chalcocite.

FIG. 11.—CALUMET AND ARIZONA MINE, BISBEE, ARIZ.

× 42.

An early stage of the alteration of chalcopyrite to secondary chalcocite.

FIG. 12.—NEVADA-DOUGLAS MINE, YERINGTON, NEV.

× 42.

A more advanced stage of the alteration of chalcopyrite to chalcocite.



FIG. 7.

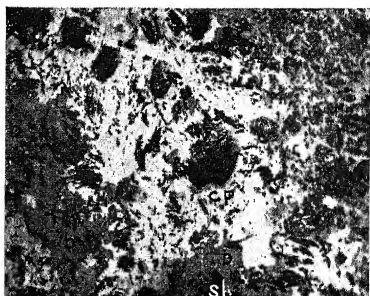


FIG. 8.

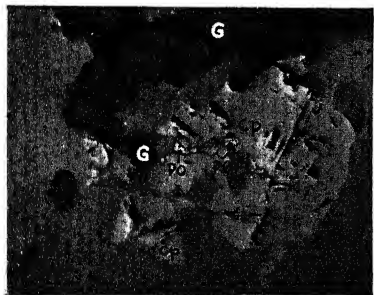


FIG. 9.

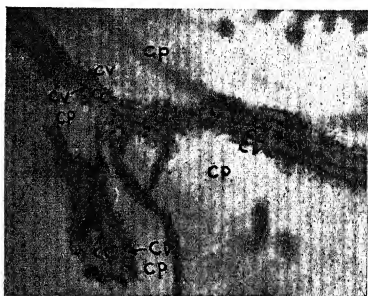


FIG. 10.



FIG. 11.

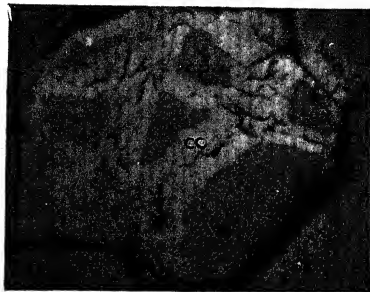


FIG. 12.

FIG. 13.—CALUMET AND ARIZONA MINE,
BISBEE, ARIZ.

× 30.

A late stage of the conversion of chalcopyrite into secondary chalcocite with the formation of intermediate halos of bornite.

FIG. 14.—LOCALITY UNKNOWN.

× 30.

Chalcopyrite altering to typical aggregates of covellite blades. This specimen is porous, and the black areas represent the cavities.

FIG. 15.—BONANZA MINE, COPPER
RIVER REGION, ALASKA.

× 30.

An unusual type of ore from this mine showing chalcopyrite secondarily after bornite, and itself altering in patches and along veinlets to chalcocite. The thin strips of bornite residual between widely-spaced cleavage-lines are plainly shown. One of these, included within the circle in the center of the photograph is shown much enlarged in Fig. 16

FIG. 16.—SAME AS FIG. 15, BUT AT A
MAGNIFICATION OF 600 DIAMETERS.

Bornite altering along its cleavages, nearly converted into chalcopyrite. A strip unaltered, possibly because it has less perfect cleavage than the surrounding bornite, is shown. A little covellite and chalcocite are developed along the edges of some of the bornite.

FIG. 17.—SPECULATOR MINE, BUTTE,
MONT.

× 36.

Pyrite, bornite, enargite, with chalcopyrite probably secondary developing from enargite, and certainly secondary developing along the cleavage lines of bornite.

FIG. 18.—CORNELIA MINE, AJO, ARIZ.

× 165.

Bornite altering along the cleavage to chalcopyrite, and around the edges to chalcocite and feathery covellite.

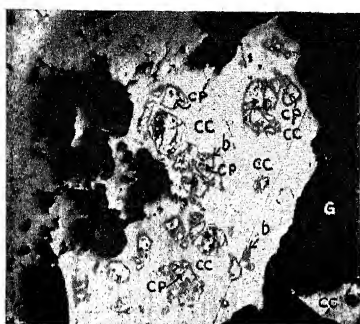


FIG. 13.

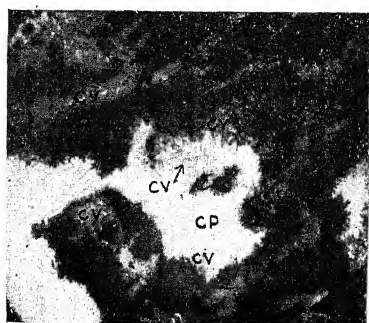


FIG. 14.

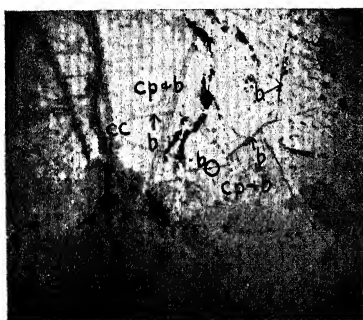


FIG. 15.

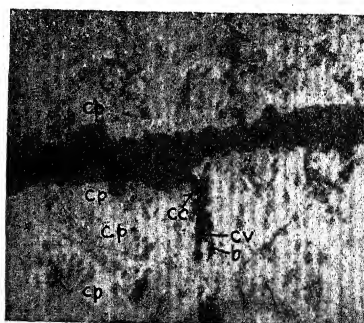


FIG. 16.



FIG. 17.

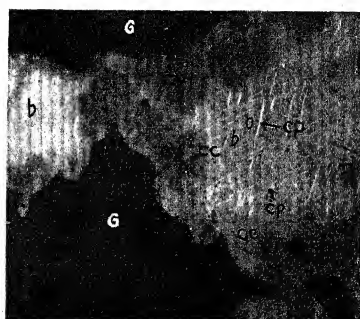


FIG. 18.

FIG. 19.—AFTERTHOUGHT MINE,
SHASTA COUNTY DISTRICT, CAL.

× 100.

Primary chalcocite, and bornite intergrown, the latter altering to secondary chalcocite, and characteristic sheaves of covellite plates, with also a little chalcopyrite. The gangue consists of prismatic crystals of barite.

FIG. 20 —UNION MEXICANA MINE, CANANEA, SONORA, MEXICO.

× 24.

Bornite, chalcopyrite, sphalerite, and tetrahedrite, showing characteristic primary structure. Specimen has been etched to bring out the cleavage of bornite. The rectangular pits developed in the bornite by this process are shown. The etching has also dissolved out a few grains of galena. The other minerals have not been attacked.

FIG. 21.—THREE R MINE, PATAGONIA,
ARIZ.

× 36.

A higher magnification of a portion of the same specimen as shown in Figs. 5 and 6. This view shows minute residual grains of pyrite, secondary chalcopyrite being altered throughout its area to bornite. The latter is cut by veinlets of chalcocite which show a typical dark central line that marks the original fracture.

FIG. 22.—NORTH BUTTE MINE, BUTTE
MONT.

× 30.

Two specimens showing pyrite and typical primary intergrowth of chalcocite and bornite. In the right hand specimen, which contains a little sphalerite also, the typically cracked and rounded nature of the pyrite is well shown.

FIG. 23.—GUILFORD MINE, GUILFORD
COUNTY, N. C.

× 36.

Shows primary intergrowth of bornite, and chalcocite partly developed micrographically. Near the bottom of the photograph, a narrow veinlet of secondary chalcocite is shown cutting the bornite but producing no effect on the primary chalcocite.

FIG. 24.—TANALYK MINE, SIBERIA.

× 36.

Primary bornite, chalcopyrite, sphalerite and galena. The bornite and chalcopyrite being altered along their boundaries with the gangue to narrow borders of chalcocite. Veinlets of the same secondary mineral are developing along fractures in the galena.

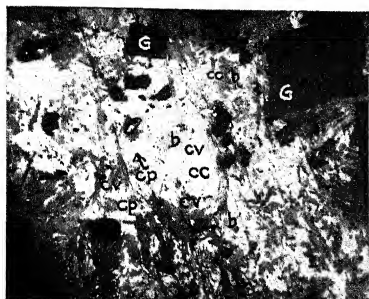


FIG. 19.

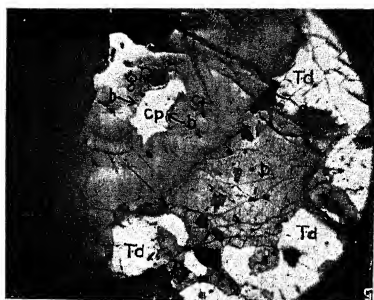


FIG. 20.

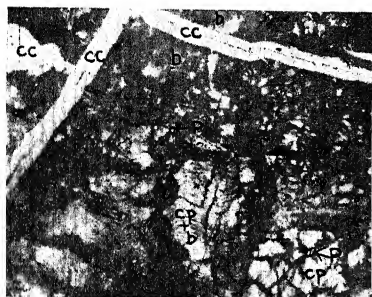


FIG. 21.

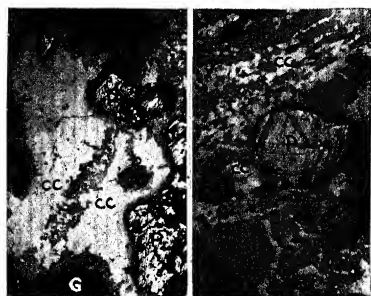


FIG. 22.

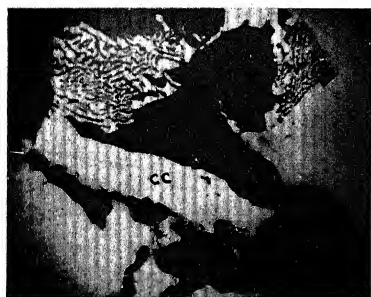


FIG. 23.

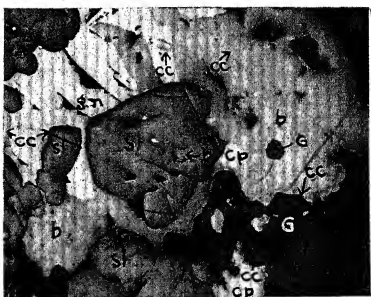


FIG. 24.

FIG. 25.—SEABOARD MINE, VIRGINIA, $\times 36$.
 FIG. 26.—CORNELIA MINE, AJO, ARIZ. $\times 36$.

Bornite altering along fractures to veinlets of secondary chalcocite, which include peculiar dendritic forms of some transparent material, probably a gangue mineral. The typical central line in the chalcocite veinlets, marking the original fractures, is well shown, also the hazy, indefinite boundary-line between the chalcocite and bornite.

Bornite with a little primary chalcopryite altering along fractures to secondary chalcocite. The same dendritic structure is shown in these veinlets as in Fig. 25, also the black medial lines, but in this case the boundary between the secondary chalcocite and the bornite is very sharp.

FIG. 27.—BONANZA MINE, COPPER RIVER DISTRICT, ALASKA. $\times 24$.
 FIG. 28.—SCHUYLER MINE, ARLINGTON, N. J. $\times 36$.

Typical massive chalcocite ore of this mine, etched to show characteristic cleavage of coarse-grained chalcocite. Regarded as an indication of the primary nature of this mineral

A specimen of chalcocite ore which has been exposed for many years on the dump of this old mine, which was worked before the Revolution. The cleavage has been developed naturally by the process of weathering, and the darker pits along the cleavage-planes represent places where the chalcocite, becoming sooty by extreme alteration, has been torn out in the process of polishing.

FIG. 29.—SYNTHETIC CHALCOCITE PRODUCED BY FUSING TOGETHER COPPER AND SULPHUR AT $1,100^{\circ}$ C. $\times 36$.
 FIG. 30.—OLD DOMINION MINE, GLOBE, ARIZ. $\times 30$.

A single large individual showing crystal-faces on the surface has been etched and beautiful cleavage is indicated—no more perfect, however, than that exhibited by the natural occurrence in Fig. 27.

Typical massive chalcocite ore formed by secondary enrichment. The etching shows innumerable cracks, the larger of which doubtless represent the position of fractures that crossed the primary ore. The smaller areas which look hazy in the photograph really exhibit the same irregular cracking on a much smaller scale, but occasionally small grains show an approach towards cleavage. The black areas represent sooty portions in the chalcocite where solution by the etching fluid has been extreme.



FIG. 25.

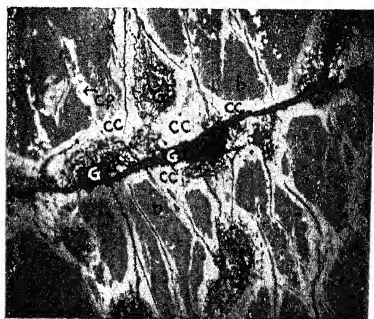


FIG. 26.

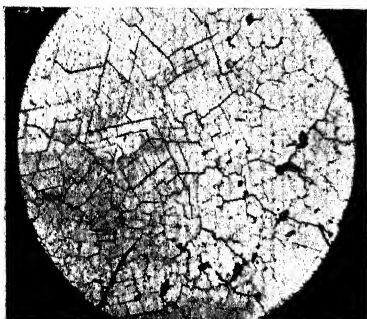


FIG. 27.

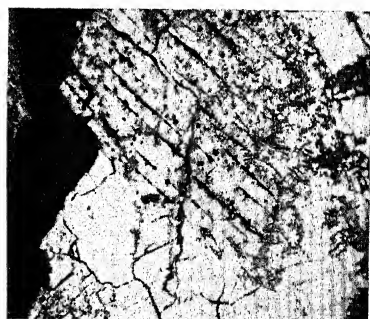


FIG. 28.

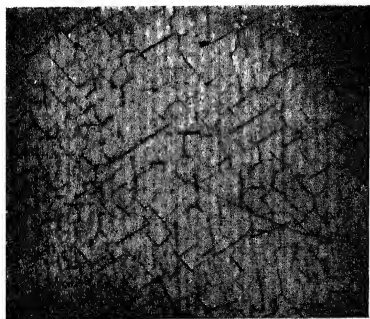


FIG. 29.



FIG. 30.

FIG. 31.—WALL MINE, VIRGILINA, VA.
× 30.

Intergrowth of primary chalcocite and bornite with micrographic or eutectic texture in the center of the field, etched to show the cleavage of the primary chalcocite and the rectangular solution-pits in the bornite. Although the photograph does not satisfactorily bring it out, the spongy mass of chalcocite is found to possess the same direction of cleavage in all its portions and is, in reality, a skeleton crystal. By etching of bornite also it can be determined that that mineral likewise in these graphic intergrowths possesses similar orientation throughout.

FIG. 32.—OLD DOMINION MINE, GLOBE
ARIZ.
× 30.

Secondary chalcocite, irregularly fractured. Along the openings thus formed oxidizing solutions have penetrated and developed native copper directly from the chalcocite. The contact between the two minerals is absolutely sharp.

FIG. 33.—COPPER QUEEN MINE, BIS-
BEE, ARIZ.
× 42.

Bornite altering to secondary chalcocite. The pattern shown on the latter is cleavage brought out by incipient etching by the acid fumes of the laboratory. The continuation of these cleavage-lines is seen penetrating into the residual cores of bornite and in reality the cleavage exhibited in the chalcocite is really bornite cleavage inherited by the secondary mineral.

FIG. 34.—LEONARD MINE, BUTTE,
MONT.
× 52.

Plates of covellite, believed to be but not certainly primary, surrounded by secondary chalcocite which is replacing primary bornite. As in Fig. 33, the chalcocite has been etched by acid fumes in the air and shows a beautiful cleavage, which in this case also penetrates into the bornite grains. This cleavage has likewise been retained pseudomorphously in the chalcocite but is really bornite cleavage.

FIG. 35.—KYSHTIM MINE, RUSSIA.
× 45.

Pyrite and sphalerite with veinlets of secondary covellite cutting the latter.

FIG. 36.—TANALYK MINE, SIBERIA.
× 42.

Chalcopyrite, tennantite, sphalerite, and galena, typical primary intergrowths. Pits in galena due to cleavage are well shown.

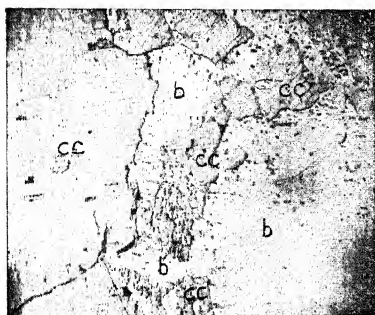


FIG. 31.

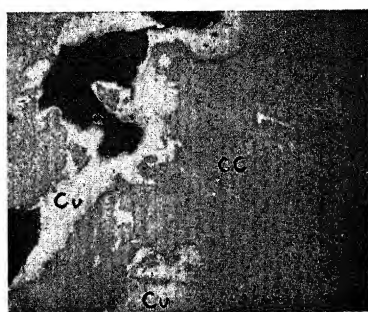


FIG. 32.



FIG. 33.

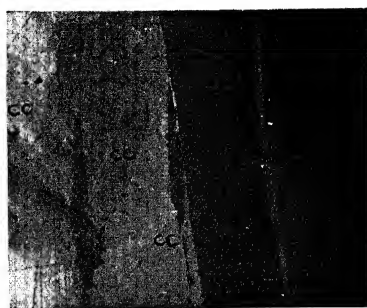


FIG. 34.



FIG. 35.

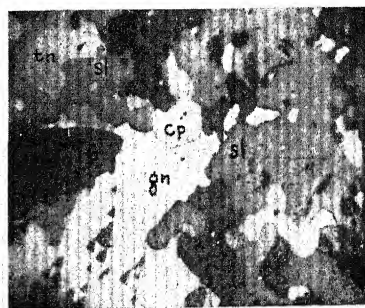


FIG. 36.

DISCUSSION.

CHAIRMAN JAMES F. KEMP:—We have been privileged in the New York Section of the Institute to hear the presentation of this paper several months ago, with illustrations projected by means of the lantern. It impressed me at the time as being one of the most important contributions which we had had presented before the Institute in some years, and I can assure our members that the paper will bear very careful perusal and study. It is also admirably illustrated by the plates which follow at the close. I feel that the American Institute of Mining Engineers could hardly let the paper pass without a word of thanks and congratulations to Mr. Graton and his associate, Mr. Murdoch, for the very able work which they have produced.

HORACE V. WINCHELL, Minneapolis, Minn.:—I feel I should add a few words of appreciation to the authors of this paper, although I have not yet had an opportunity to examine it carefully. I have only glanced over it hastily, yet I am in a position to appreciate somewhat the magnitude of the investigation and the results derived. My mind goes back over the last ten or twelve years—perhaps I might say fifteen years—to the time when the possibility of such a thing as secondary sulphide enrichment was discussed actively and almost without interruption, in Butte, with Clarence King, Dr. Raymond, and several other eminent geologists who came out there to help me study the ore deposits in connection with some of the early mining litigation.

The question of the possibility of the formation of copper glance by descending waters was often raised, and as frequently turned down; yet it would not stay down. It was suggested that the paragenesis of the minerals, particularly the chalcocite, its relation to the recent fractures, and its abundance in certain portions of the mine, made it evident that it was often the latest to be introduced into the veins. The investigations reached the point of establishing what was probably the first laboratory for the study of the origin of secondary sulphide ores in Butte, in 1899, and in the few months during which the laboratory was operated we caught the first clue, and succeeded in producing chalcocite the latter part of that year.

The question as to the reducing agent which was able to

form cuprous sulphide from the sulphate solutions found in the veins was the first obstacle which we met. This we could not at first determine. All the writers theretofore had assumed that organic matter must be the reducing agent. We experimented with organic matter. We found in the fire zone in the Anaconda mine, where the mine fire has not yet been extinguished, that the sulphate solutions had been affected by organic matter and copper was deposited in the timbers, and it was native copper every time. There was the result of organic action. We found that tar ropes left in the bottom of the shaft in the mine-waters, which were green with copper sulphate and iron sulphate, likewise precipitated native copper. We had instances of accumulations of mossy copper formed in that way upon these tar ropes.

The question of the reducing agent then led to the investigation as to the amount of carbon monoxide, perhaps in the shape of gases, included in the rocks, and we made some determinations of this, and again we were unable to get any chalcocite. We investigated the possibility of the dissociation of water and the effect of hydrogen, and again obtained native copper; and we tried hydrogen sulphide, only to obtain a mixture of amorphous cupric and cuprous precipitates; and finally it was discovered that in the processes of oxidation—the natural oxidation of these sulphides—the superfluous molecule of sulphur is again carried into solution as SO_2 , and that a solution of SO_2 produced the desired reducing effect; and since that early experiment it has been recognized that it is in association with the processes of oxidation and reduction that chalcocite is formed.

In that connection I am reminded of the conditions which I found in Alaska, where, in the beds of the streams, in the ice cold water, native copper nuggets are found incrustated with the secondary sulphide mineral, chalcocite, and some bornite, in the gravel immersed in the cold water and assembled with other heavy minerals in the bottoms of the streams. There is a case where it would seem that oxidation must have been sufficiently rapid almost to prevent such possible formation of sulphides. There is a case of the deposition of chalcocite, not within a vein, not in the earth's crust, not at high temperature, not excluded from oxygen—and I explained it as follows:

Associated with the native copper nuggets I found smaller pebbles of pyrite, and the pyrite must have been oxidizing and must have furnished sulphuric acid sufficient to take copper from the copper nuggets, and to furnish the sulphurous acid which reduced the cupric sulphate, first to cuprous sulphate, then to cuprous sulphide, and finally deposited the latter, incrusting and surrounding the native copper nuggets, within 10 ft. of open daylight. Instances of this action are abundant and of common occurrence; and here, I should suppose, we have an example of the direct formation of chalcocite, and not by the replacement of any copper sulphide mineral.

I have recently been to Butte, and visited the lower levels, and the conditions there are surprising. At a depth of 2,800 ft. chalcocite is still found in great abundance, and there also are found enargite crystals in the process of decay, breaking down and completely, or almost completely, gone. There is an example of the theory presented by this paper, that copper ores are formed rapidly; but, though the deposit as a whole may have been due primarily largely to a rather rapid process of deposition, still, we do find at that great depth, indications of gradual changes within the vein, and slow transformation from one copper mineral to another.

There is no doubt, I think, that a large amount of chalcocite in Butte is primary. And I may say that we do not find in connection with this copper glance the complementary mineralization which I should expect to find, but we have found what I suggested in my first little paper on the subject, a little specular hematite in connection with some of the chalcocite from alkaline solution.

I am glad to see that this investigation is to be carried on in many camps through one pair of eyes. Certain it is that in no other way can so valuable an opinion be formed by any observer. No student of the subject can use the observations and descriptions of others and arrive at the same certainty of conclusion, the same logical, consistent explanation, as he can reach by basing his conclusions on what he himself sees. I congratulate Mr. Graton and his co-author upon this very excellent contribution.

PROF. ALFRED J. MOSES, New York, N. Y.:—I have been

interested in this new method of testing used by Mr. Graton, and have accumulated what literature I could on the subject. I have not found that very many tests have been described, and in studying the literature it has seemed to me that conclusive proof, ordinarily speaking, of the identity of the species is lacking. We have only conclusions based on tests the value of which has not yet been thoroughly worked out.

The identity of a species based on tests of color, etching, shape, etc., may probably be correctly obtained from a polished section by reflected light; but since there are various species that seem to be likely to give the same results with the tests hitherto described, it seems to me that until we can collect data on these other opaque minerals, such as the silver sulphides of the same color as the copper sulphides, and unless more tests are used than have been described as yet, we hardly know what we have been dealing with. We may have, for instance, something which is apparently a chalcocite, but so far as any tests on record show it might have been argentite, or stephanite, or some other gray to black mineral with similar forms, etching tests, and color. If a micro-chemical test is added for copper, chalcocite of course becomes more probable. I am not criticizing the conclusions of Mr. Graton, but calling attention to the fact that the tests upon which he bases his conclusions have not been stated, and there should be some way of getting this information and adding to it, so as to secure a definite means of identifying these and other opaque minerals with the microscope.

I do not know how far Mr. Graton may have devised a conclusive series of tests, but hitherto the conclusions seem to have been based on a very small number of particular characters, which have been proved to be obtainable from one mineral and not to be obtainable from some second mineral, but may very possibly be obtained from a dozen others not tested.

I am very anxious to have mineralogists and chemists take up this matter and get the facts together so that the conclusions may be justified. I have no reason to say that the conclusions which are given are not correct. I have very little doubt they are correct, but I think there is an unfortunate absence of statements showing how the minerals were identified. It is that which interests me as a mineralogist.

E. S. BASTIN, Washington, D. C.:—I think that the paper by Messrs. Graton and Murdoch forms a beautiful example of the growing importance of the study of polished sections of ore. Such studies are coming to be just as important to the economic geologist as the study of the thin sections of rocks has become to the petrographer. The development of criteria for identifying the various opaque minerals will go hand in hand with the expansion of these studies. The attention which has been given to microscopical studies has opened up many new fields in the study of ore deposits. It is surprising, when one commences such studies, to see how an ore which to the naked eye looks to be all of one period of formation, will in the polished section resolve itself into an example of a very complex history.

Mr. Graton has worked out some interesting results with the copper ores. Work that has been going on at the U. S. Geological Survey²³ has produced somewhat similar results with silver ores, in which secondary polybasite and other secondary silver minerals occur. In the hand specimen these silver minerals might be interpreted as original constituents, but in polished sections it is at once seen that they are metasomatic replacements of the original ore minerals.

It is usual in considering secondary enrichment in silver to think of the secondary silver minerals as occurring in cracks in the original ore. In some of the silver ore from Gilpin county, Colo., and Nelhart, Mont., they do occur to some extent in fractures, but they occur in as great or greater abundance in the solid ore, as replacements of quartz, galena, and other original ore minerals. The cracks that admitted the solutions were so minute that they are hardly recognizable under the microscope, and yet they have sufficed to let the solutions in and permit a gradual replacement of the original minerals.

Mr. Graton referred to his hesitancy in writing any chemical equations from such replacement reactions, and I am inclined to think that such reluctance is well founded, as Mr. Lindgren²⁴ has pointed out. In the case of such reactions as those exhib-

²³ Metasomatism in Downward Sulphide Enrichment, E. S. Bastin, *Economic Geology*, vol. viii., No. 1, pp. 51 to 63 (Jan., 1913).

²⁴ The Nature of Replacement, W. Lindgren, *Economic Geology*, vol. vii., No. 6, pp. 534 to 535 (Sept., 1912).

ited by the speaker earlier in this meeting, that take place in a test tube under static conditions, a chemical equation can correctly express what takes place. You have a certain amount of material to start with, and that must be exactly balanced by the end products of the reactions. The replacements that take place in nature in secondary enrichment differ from these in that they are accomplished by circulating solutions. These solutions are constantly supplying new material. As a result, chemical balance not only need not exist, but in most cases at least it cannot exist, and it is impossible to write a chemical equation that will correctly express the changes involved.

Another point which is especially interesting to me is Mr. Graton's conclusion that the phenomena of secondary enrichment are largely phenomena of oxidation. It is rather interesting to note that a number of observers of these processes are working to the same general conclusion. The phenomena that were shown by the speaker's experiments on silver solutions, exhibited earlier in this meeting, were also shown to be phenomena of oxidation. Although we generally look upon the phenomena of oxidation in ore deposits as confined mainly to the zone above the water table where the oxygen was supplied by the air, we appear to be accumulating evidence that the phenomena of oxidation may extend to depths far below the ground-water level. Secondary chalcocite develops at very considerable depth and secondary enrichment generally may go to hundreds of feet below the ground-water level. In Gilpin county, Colo., silver ores in some mines showing secondary enrichment extend at least 700 or 800 ft. below the ground-water level. If these are oxidation processes, where does the oxygen come from? There are two possible sources for it. It may come from the air that is carried down in the water; but that seems to be an inadequate source for the amount of oxygen necessary, and it may be well to emphasize the point that these experiments show water itself to be capable of furnishing oxygen for certain reactions involving the deposition of the precious metals. I should not be surprised, therefore, if future investigations confirmed the suggestion that, in the solutions producing secondary sulphide enrichment, we have in the water itself an agent for oxidation at considerable depths below the so-called oxidized zone.

THE CHAIRMAN :—Are there any further remarks? If not, we will call upon Mr. Graton to close the discussion.

L. C. GRATON :—I was especially interested in those portions of Dr. Bastin's valuable paper in which parts of his field and of our field overlap. Particularly interesting is his conclusion that the hydrolysis of water alone might effect oxidation of the primary ore, and in that way precipitation of the secondary silver. That, if true, readily explains the occurrence of secondary ores below the ground-water level. It seems to me, however, that it does not quite explain, or, rather, it may possibly complicate explanation of the fact, that these secondary ores, though they may occur somewhat below the ground-water level, do not in every case extend to the very bottom of the ore body. If water alone is enough to effect oxidation and these processes go on so rapidly under proper conditions, why is not the whole ore body oxidized to the extent of precipitation of secondary silver? In regard to silver, I will not attempt to say, because my experience at first hand is far too meager; and as regards copper, I hesitate to say, because I want to keep as open a mind as possible for an investigation of secondary enrichment soon to be undertaken. It is a fact, however, that copper ores of secondary origin undoubtedly occur below the ground-water level; but there is the possibility that, since these secondary ores were deposited, the ground-water level may have risen. In certain places this can be proved; in numerous other cases the proof does not yet exist.

Regarding Dr. Bastin's experiments, we also have found rarely in silver-bearing ores that had been enriched, native silver precipitated in cracks in the chalcocite. We had not reached any possible explanation, but I believe that Dr. Bastin's recent work illuminates such an occurrence and that, without question, his explanation applies directly there. We have found very little argentite in connection with such native silver, although the ore might be richer in silver than seemed to be accounted for by the native silver present, and the only conclusion we have reached as yet—that is, our best guess—is that the chalcocite itself is argentiferous.

That brings up the question Professor Moses mentioned, namely, the identification of the various mineral species, and I

am fully in accord with him as to the difficulty and the necessity for care in this. We found in our work that the designation "unknown" was coming with such annoying frequency that we felt we had to do something about it, and for a time we left the copper ores, as such, altogether, and started to work on known specimens of carefully determined mineral varieties. We have now gone over practically the whole list of opaque minerals which have any importance in ore deposits. There are comparatively few other rare species, which are known as mineral curiosities, which we have not worked with. We have tried to develop a systematic scheme of micro-chemical analysis. We found at the outset that we are limited, or we think we are limited, to that type of chemical reaction which produces a precipitate of characteristic color. We treat the grain of doubtful or unknown identity with minute drops of various reagents in order and observe the effect. We have been able to establish groups and sub-groups corresponding to the procedure in ordinary qualitative analysis, and by applying the appropriate reagents in proper sequence we can generally sift down to a determination of what the mineral is. This scheme is not yet perfected, and there are some twenty specimens we have studied in which there are minute particles which we still have to label "unknown." Whether we shall be able in the course of another year or two to round these up, I do not know.

Professor Moses was right in saying he did not find the evidence of our conclusions as to mineral determination in our paper. We felt that we had made it pretty long already, and deliberately refrained from saying anything about either the appearance or identification of the mineral species under the microscope, or the methods of manipulation. Mr. Murdoch is working especially upon that subject, and I trust will present as a doctor's thesis in the near future some fairly comprehensive statement of the methods of determination of the minerals in polished sections.

I was very much interested in the description of those copper pebbles coated with bornite and chalcocite referred to by Mr. Winchell. I believe he described these briefly some months ago, and I have since been hoping that the occasion might arise to ask him for some of that material for study.

The agency of sulphur dioxide is undoubtedly important in the precipitation of these secondary sulphides, and some recent work of Prof. W. H. Emmons indicates that hydrogen sulphide is also important. Our own work up to the present has made us feel that, whatever the exact chemistry of the reaction is, probably in the majority of cases, if those gases are the essential reagents, they are active only in the nascent condition, or at least immediately at the point where they are generated. If you oxidize pyrite and produce sulphur dioxide or hydrogen sulphide, then shoot it off through a crack up a level or two and there mix it with copper sulphate that has come down through a fracture from a level or two above, you may be able to precipitate chalcocite, but I doubt if much secondary chalcocite is formed in that long-range way. If you get the copper sulphate right down against the pyrite maybe it is SO_2 or H_2S that does the work. In fact, is it not likely that it is this very copper sulphate from which the chalcocite is formed that oxidizes the primary sulphide ore? I am inclined to think that these copper pebbles represent a combination of conditions that are not really typical of most enrichments.

The nature of the chalcocite at Butte is regarded generally as a very important matter. Of course, there has been a growing belief in very recent years that a considerable part, at least, of the deep-seated chalcocite of that camp is primary. We have recently had, through the courtesy of Mr. Sales, Geologist for the Anaconda Co., a very important and carefully chosen set of materials from various horizons and of various significance, and we find, so far as our information permits us to decide, an abundance of primary chalcocite. We find typical graphic intergrowths of bornite and chalcocite, and we also found intergrowths of somewhat similar nature of bornite and enargite, which we can regard only as primary. We also find, however, at various places through the entire horizon, so far as our specimens enable us to judge, that there is some secondary chalcocite; sometimes it is prominent, sometimes it is very subordinate. Simply on the basis of what little we know about the Butte occurrences, I should be inclined to consider the enargite crystals that are broken up into something else, which Mr. Winchell described, as being one phase of that enrichment process, and not a reworking of the original ore by ascending

solutions. We found, it is true, two generations of enargite in the same ores, but, so far as we can judge, the enargite in both cases belongs to the same general period of deposition, and the idea that some have entertained, that the enargite is of a distinctly later period—a second epoch of primary ore mineralization—has not been illustrated, at least in the material we have studied.²⁵

HORACE V. WINCHELL:—The process of secondary sulphide formation in the oxidized zone can be observed in almost every mine in the West producing copper glance, and the place where you can see it, and can know it, and can prove it, is, as Mr. Graton says, “right there.” There is your primary sulphide, or pyrite, and there is your chalcocite formed upon it; but how about those cases where the vein has been opened rapidly—where, behind the main quartz vein in the granite, fissures have been formed later, and there is not a particle of pyrite or any other primary sulphide, but the veins are full of solid chalcocite? There must have been a deposition of chalcocite through some reducing action other than that of the sulphide which contributed the copper and the sulphide required for that mineralization. The streaks running out many feet into the fresh, pure, almost entirely undecomposed granite, into the hanging-wall and the foot-wall of the Anaconda vein and other veins in Butte, undoubtedly contain such chalcocite, which in my opinion is secondary, because it is surely so in the upper levels, and it is what every one is now accustomed to consider secondary chalcocite, and I cannot see how it could have been formed there unless this reducing agent penetrated into the fissures and there had some agency in its formation.

L. C. GRATON:—That point is most important, it seems to me, and I have, perhaps, a bit of light to shed upon it. I happened to be in Butte shortly after the Anaconda vein was cut on the 2,400 level, if I remember rightly. My recollection is that within a few feet of that vein was cut a stringer several inches in width of massive chalcocite ore in kaolin. I presume that Mr. Winchell refers to that kind of chalcocite occurrence.

²⁵ Additional material from Butte received since these remarks were made indicates that this statement may have been unduly positive.

MR. WINCHELL:—I said, in perfectly fresh granite.

MR. GRATON:—In fresh granite—well, I don't know about that. I remember later talking over this occurrence with Mr. S. F. Emmons, and we both felt that here was a case where secondary chalcocite had been deposited, not by sulphide material, but by the kaolin, by the process called adsorption,—that by some hocus pocus way the silicate had interacted with the copper sulphate solution and precipitated sulphide. I had saved a piece of that ore, and a couple of years ago made a polished section of it, and I was quite surprised to find minute pyrite remnants in it that probably never would have been detected in the ordinary sort of examination, even with a hand lens. It looked very much like the material illustrated in Fig. 4 of our paper, except that, in this particular specimen which I took along with me from Butte, there was not so much pyrite as is shown in this Fig. 4. What may be the nature of the chalcocite in fresh granite I do not know, for I have never seen it.

THE CHAIRMAN:—I was reminded, when Mr. Winchell was speaking, of the great prejudice against sulphides, as illustrated in this way: We remember that the mines at Ducktown were active in the 40's and 50's. The miners took out the rich black ores near the water level and shipped them by way of Mobile to England. In my college days, as a student, we were taught that these black, sooty ores were the black oxide of copper. Not until years afterward, when Mr. Weed touched a blow-pipe to them, was it discovered that they were chalcocites. They were brought up as an illustration of the rather unusual occurrence of black copper in the Ducktown mines, but, when one has studied this mineral, he never hesitates to say, at first sight, that it is chalcocite.

C. Q. PAYNE, New York, N. Y. (communication to the Secretary *):—Professor Graton has presented the transformations which take place among the copper sulphide ores by secondary alteration so clearly, and accompanies his explanations with such frank discussion of his own results, that one is predisposed to accept his conclusions in advance.

After reading his paper one acquires a fresh insight and interest in the mineral kingdom, and those of us who have not already acquired a reflecting metallurgical microscope will be strongly inclined to do so.

Since Professor Graton has referred to the zinc mineral sphalerite in his conclusions, I would like to mention a very striking case of the secondary alteration of sphalerite which came under my observation several years ago, and which supports Professor Graton's broad conclusion that secondary enrichment is essentially an oxidizing process.

In the Horn Silver mine at Frisco, Utah, there is a considerable deposit of sphalerite, which has been formed in andesite close to a limestone hanging-wall. The mine has been opened by driving levels in the usual way in the process of mining the lead, silver, and copper ores which have occurred in more or less close association with the zinc ore. Owing to associated minerals which present difficulties in its ore treatment, the zinc ore has heretofore been left among the ore reserves. In several exploratory drifts which have been run under the main body of the zinc ore there occur local patches or bunches of goslarite, the hydrated sulphate of zinc. This mineral here forms hair-like filaments of a snow-white color, in places a foot or two long, and presents a very striking and picturesque appearance. Its comparatively rapid growth seems to bring it almost into relation with the vegetable kingdom. Being soluble in water, goslarite could only exist in a dry mine, and yet sufficient moisture evidently exists in the country-rock to permit a slow downward movement of cold, acid water, which, acting on the sphalerite, has caused the formation of goslarite on the roof and sides of abandoned drifts in certain places.

This occurrence seems to be one of the few cases in which no increase of scale, or magnification, is required to permit the geological relation between the two minerals to be observed and clearly understood.

The India Mica Industry.

BY A. FAISON DIXON, NEW YORK, N. Y.

(New York Meeting, February, 1913)

INTRODUCTION.

IN India the production of mica, which in other countries is of very minor importance, is one of the staple, long established industries, and ranks high in the statistics of mineral products. Nearly two-thirds of the mica of commerce comes from India. The Indians have used mica for glazing and decorative purposes through many centuries. About 14,000 men are employed in and about the mines, and the indigenous and primitive mode of mining is still followed; but, by reason of the exhaustion of the surface supply hitherto won without intelligence or capital, it will not be many years before the present interesting and casual methods will be replaced by a less archaic and more profitable system. India on the whole has little mineral wealth, but the unique resources of its mica-fields, cheap labor, and favorable natural conditions render it unlikely that any other country can ever more than temporarily contest her leadership in the production of this one minor but essential mineral.

Mica plays a small but indispensable rôle in a variety of fields of dissimilar application. Its use and value depend upon characteristics which different kinds of mica have in varying degrees, each kind being adapted to its own special purpose. Its well-known characteristics of transparency, flexibility, elasticity, toughness, high dielectric strength, perfect flatness, ease of splitting into thin films, ability to stand high heat and sudden changes in temperature, give it value in many industries. No other substance, natural or artificial, has the same combination of qualities.

The oldest use of mica is in glazing. By reason of the improvements in annealing glass, this is not now of great importance, but mica is still employed for the covers of ikon images in Russian churches, the windows of anthracite stoves, compass

cards, mariners' compasses, windows in the turrets of battle-ships, lamp-chimneys, and other places where glass would speedily be broken, or light weight is required. The diaphragms of phonographs and of some telephones are of mica, and there are other uses of mica dependent on its peculiar elastic properties in a thin sheet. The great use of mica at the present time, however, is in the electric industry as an insulator. It is indispensable in dynamo-electric machinery, and the enormous increase of such machinery is the cause of the great rise in price of all kinds of mica. Large motors and dynamos use large perfect pieces or built-up sheets, and the little magnetos make a market for the small sizes. Fuse-plugs, lightning-arresters, and condenser-plates for wireless telegraphy also require mica as an insulating material.

All the mica of commerce is either muscovite or phlogopite; India produces only the former. The mineralogical and chemical characteristics of the minerals of the mica group and the theory of the genesis of mica deposits are given in detail in some recent publications¹ and need not detain us here.

The valuable muscovite of India varies greatly in different mines in hardness, clearness, transparency, etc. The characteristics for which the consumer looks can generally be found in some one of the grades of Indian mica.

There has been a sporadic production in the Nilgeri Hills, Bombay, and in Ragputana; but the two important fields are the Hazaribagh district of the province of Behar and Orisar, and the Nellore district, Madras Presidency. The former furnishes two-thirds of the Indian production.

THE HAZARIBAGH DISTRICT.

Mining in Hazaribagh centers around Kodarma, a village of a few hundred inhabitants, with a dozen or more Europeans living nearby, situated 4 miles from the East India railroad, and 160 miles from Calcutta. The elevation (1,000 ft.) moderates slightly the intense heat of the Indian summer. The year is divided into three periods, giving pleasant cool weather from

¹ The Mica Deposits of India, by Sir Thomas Holland. *Memoirs of the Geological Survey of India*, vol. xxxiv., part 2 (1901). *Mica and the Mica Industry*, by George Wetmore Colles (1906). *Mica, Its Occurrence, Exploitations, and Uses*, by Hugh S. de Schmid (1912).

November to March, extremely hot weather from March to June or July, and the "rains" from July to October, during which season the air is cooler, but very muggy. The rainfall (160 in. per annum) occurs almost entirely during the "rains." The country is rolling with low rounded hills of gneissic granite, which are weathered into piles of ellipsoidal masses of bare rock by the action of the sun and weather.

Geology.

The important mica-bearing area is north of Kodarma, and extends in a belt east and west about 20 miles long and 7 or 8 miles wide; but there are several large isolated deposits scattered outside of this area for 40 miles further east. Most of the mines are accessible by carts except during the rains.

The two principal rocks of the district are schist and the "dome-gneiss." The schist is the country-rock of the mica-bearing pegmatite veins, and covers the larger part of the area. It is composed of highly contorted bands of various mineralogical composition grading from quartzite to one largely consisting of sericite. The dome-gneiss, which forms the hills mentioned above, is composed chiefly of quartz and microcline, with some oligoclase and biotite, occasional hornblende, and minute crystals of sphene, apatite, and zircon. It generally shows a porphyritic structure. No pegmatite veins cut the gneiss.

The Pegmatite Veins.

The pegmatite veins generally follow the bedding of the schist, which strikes most frequently in a northeasterly direction. The great diversity in these mica-bearing veins makes it impracticable to give any very useful statement of characteristics common to them all. Valuable mica is found scattered through veins of solid quartz or feldspar, in kaolin, and in pockets surrounded by what is little more than silicified schist. The mica presents every variety, from the soft, silvery white, which competes with Canadian phlogopite, to the hard, clear "ruby," which rings like a bar of metal on being tapped. Mica with slight stains of iron oxide, or with occasional black spots, is the most common. These stains and spots are generally between the laminae, and if not too numerous can often be removed by careful splitting. Some is densely stained, in thick sheets look-

ing like biotite. It has a ready sale, but at a lower price. In many cases, the mica from a given mine has some peculiarity which makes it easily distinguishable, such as a few large black spots, scattered dust-like specks, dark streaks along crystallographic axes, a wavy or flat surface, or some combination of these characteristics. The average size of the mica plates also changes from place to place. The rock is generally little altered on the surface. I have noticed small crystals of unaltered mica projecting from a ledge of pegmatite. In mines which contain kaolin, the schist wall-rock is generally unaltered, and I have heard of no case of getting through the kaolin into unaltered ground. In only one place have I seen graphitic granite—a small prospect-pit in the NW. corner of the government forest.

The most common type of deposit is a well-defined continuous vein of quartz and feldspar with a fairly regular strike outcropping along the top of low ridges. A vein may pinch to a few inches, expand to 30 ft. or more in width, or break up into smaller veins, which join together again; but on the whole the veins of pegmatite are fairly straight and regular. The mica content, however, is far from regular; each occurrence has its peculiarities in the relation of the mica to the other constituents and to the vein itself. The most common type of workable deposit is massive quartz filling one side or the middle of the vein, with solid feldspar on one or both sides. Generally the feldspar only is mined; not because it is necessarily richer, although I think it generally is so, but because it is softer and easier to work. Little ground has been worked that has not continuously shown another "book"—*i. e.*, marketable crystal—of mica when one has been taken out. Old diggings, therefore, invariably show what was formerly mica-bearing ground. Along the strike some deposits show regular continuity in mica content for a quarter of a mile or more; but most of them have been mined only for 100 ft. or so. Those most familiar with the mica-mines of Kodarma are least dogmatic as to what constitutes a good indication of mica.

Aside from a little tourmaline, biotite, and an occasional garnet or beryl, the amount of accessory minerals is small, although many have been identified. Considering the extent and diversity of the pegmatite veins, it is remarkable how few minerals besides those mentioned are ordinarily seen. The

mica occasionally shows crystal faces, but the edges are more often rough. Frequently great masses of pure mica weighing a ton or more, yet containing no material of commercial value, are found. In fact, too large a percentage of mica in the rock is considered a bad sign, because the crystals interfere with each other, and, as the miners say, "the books do not make up." The serviceable mica is generally found in books scattered through feldspar or quartz like plums in a pudding.

There are practically no mine-maps, and of the extensively worked deposits only a small portion of the workings is accessible. The surface of nearly every mine is covered with waste, which makes it difficult to get accurate data concerning the relation of the pegmatite to the country-rock or the mica to the pegmatite. The pegmatite masses, as a rule, follow the plane of foliation of the schist. Where the pegmatite swells out into lenticular masses, the folia of the schist follow the sides of the pegmatite as if they were pushed aside by an entering wedge. An example of this is at the Phutlai Bogda mines, near Debore.

Veins are sometimes displaced along the strike of a cross-fault. These faults may show slickensides and a clay gouge, and contain no mica of value. At other places the pegmatite intrusion seems to have followed an earlier fault-plane for a short distance; and the most valuable mica may be at the junction of two such planes of weakness. This seems to be the case at the Boria mines, near Debore.

At the Dugi mine, Phutlai, the old workings extend continuously in a straight line N. 40° W. for more than 3,000 ft., averaging about 30 ft. in depth. The vein is feldspar, with a little quartz and frequent horses of schist, which break it up occasionally into what appear to be several veins. It is mined to a width of from 3 to 5 ft., and appears to have a uniform content of medium-sized spotted mica.

At the Pania works, near Dumohanch, the mica is irregularly distributed, often in masses of large, fine books, in the silicified wall-rock of a small irregular quartz vein.

The Lugunwa and Torna mines, near Kodarma, two of the largest and most prolific mines in the district, present a type which appears to be characteristic of many of the more valuable deposits. Both mines are on the same vein, which has been developed to a depth of 120 ft. on its dip of 60°. The vein is

from 4 to 6 ft. wide, fairly regular, and consists chiefly of feldspar. Slickensides on the walls give evidence of movement subsequent to the vein formation. There is some clay gouge, which occasionally contains marketable mica. The books of mica are scattered irregularly through the feldspar, with no apparent orientation to each other or to the sides of the veins. Some good books are found imbedded in the wall-rock. The proportion of mica to vein material is not apparently as high as in many far less valuable deposits. Practically all of the vein is worth mining, except for occasional bosses of quartz. The proportion of mica brought up to go to the storehouse depends largely upon the care shown by the miners in sorting. About one-eighth of the vein material is mica, of which one-half comes to the surface as rough mica. The mica here averages fairly large.

The Dormacola mine, near Bishampur, illustrates another common type of deposit. The vein here is from 6 to 15 ft. wide, the center is mainly massive quartz, and on the walls is feldspar from 2 to 3 ft. thick. It has been worked along the strike for about 300 yards in a straight line, the feldspar only being mined.

Method of Mining.

Mining, except that dynamite is used, still follows the time-honored methods developed by the Indians before the Europeans came. A book of mica, found on the surface, was taken out; if another could be seen it also was gouged out. As the hole grew deeper, women and children formed a line and passed up waste and water to the surface. If the ground appeared rich on the side of this well-like burrow no lateral stoping was done, but another meandering hole was started near the surface and carried down near the first. If a deposit proved wide or the sides began to cave, an open cut was made and the waste heaped on either side in total disregard of drainage or the possible burying of a part of the deposit. Fig. 1 shows an adit being driven through such an old waste heap to get to the bottom of an old mine, and Fig. 2 shows a shaft sunk through caved workings at the Panya mine.

The natives can work in the most confined space and fetid air, and if left to themselves never think of straightening or

enlarging their burrows unless a book of mica tempts them. White control, for the most part, has done little more than to systematize the old method. In one mine under European control, which is about 100 ft. deep, there are 87 men, women and children, who pump out water and pass up in buckets the waste made by 8 men who are actually mining. Three or four years ago, before hand-pumps were used, twice that number would have been required for bailing. Such a simple mechanical appliance as a well-sweep or a windlass is rare; where hand-pumps are used the acme of mechanical triumph is reached. I saw one or two pulsometer pumps at the mines; but they were not suitable for the lift required and were idle. Some of these holes have been put down 300 ft., but I believe no mine is being worked at present which exceeds 100 ft. in vertical depth, the government mine-inspectors having stopped the work in the deeper mines. Vertical shafts are a rarity, and though the country is well adapted to short drainage-adits one is rarely seen. Ten years ago the Indian Mica Co. attempted systematic mining methods at the Bendi mines. The enterprise was commercially unsuccessful, apparently by reason of the expenditure of all the capital on the too elaborate equipment of a mine which unfortunately proved to be mediocre. Various proposals have been made since to mine in a more rational way, but nothing has yet been accomplished.²

Tata, Sons & Co., of Bombay, with J. F. Podger, manager, are now beginning to open along modern lines several mines on their property.

Until recently something could be said in favor of not abandoning the old-style mining. Money was made by it; little capital was required; no work was done that did not immediately produce mica; the mines for the most part were shallow—from 15 to 30 ft.; when it became expensive to work one at increasing depth, it could be abandoned for another equally good. Almost no expensive European supervision was required. The unlucky experiment at Bendi deterred others from making an attempt. But the conditions are now changed. No more mica can be found near the surface; there is very keen

² See for a discussion of this subject, *The Kodarma Mica Mines and Mining*, by A. A. C. Dickson, *Transactions of the Mining and Geological Institute of India*, vol. iii., part 1 (1909).

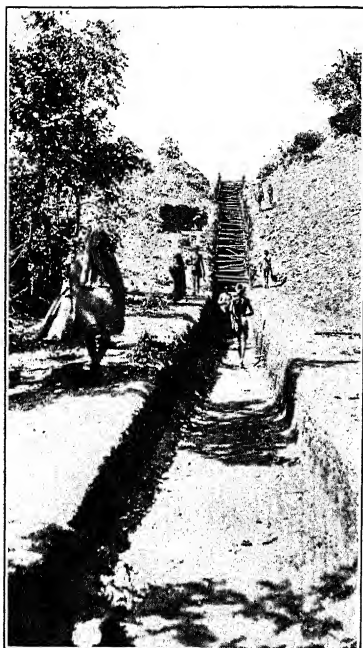


FIG. 2. — SHAFT SUNK THROUGH
CAVED WORKINGS, PANYA MINE,
KODARMA.

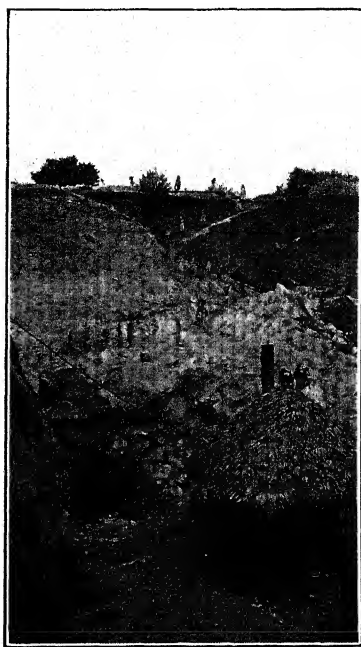




FIG. 5.—MICA CUTTERS AT WORK, KODARMA.

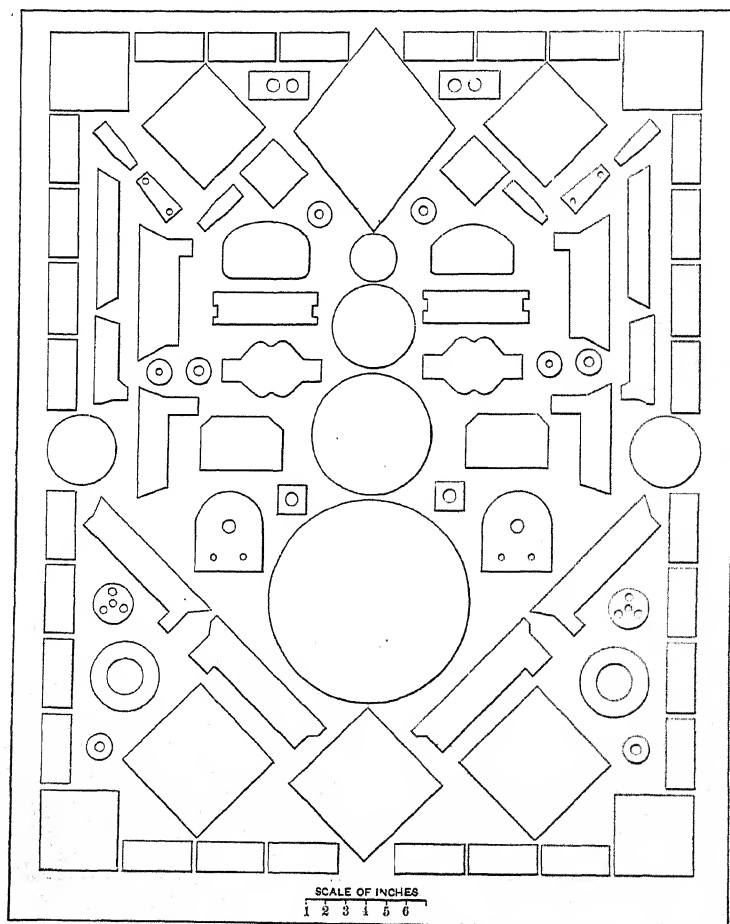


FIG. 6.—COMMON FORMS OF MICA PREPARED FOR USE.

competition for land; production has not been maintained, even at advancing prices; old mines must be reopened; the richest are naturally the deepest; they have for the most part caved and can in no case be opened successfully by the old methods. Systematic mining is thus forced upon the industry, and will doubtless benefit it in every way.

At present the little mines are scattered over many miles of jungle, often several hours' walk from the nearest village; mica in large sheets may be worth several shillings a pound; wages are low and a pound of mica may represent the value of a man's weekly pay. It is next to impossible to control effectively the miners, or their foremen, whose salaries are likewise very small. There is a ready market for any mica in all the nearby bazars; the Hindoos are no more moral than miners elsewhere; and hence it is hard to tell what portion of the mica mined reaches the storehouse. But the fact is illuminating that much more mica is exported from India than is reported in the mining returns. The concentration of work in a few places would certainly tend to check theft; but the management must in any case trust to subordinates, and there is no chance of the I. D. B. type of law being enacted in India. Each manager to whom I have spoken believes that the others lose much of their mica on the way from the mines to the storehouse, but thinks his own men would not dare to steal.

Some writers have said that much mica is destroyed under the present method of mining. Drilling and blasting at the bottom of a small hole, in a wet and dirty working-face, and with a very poor light, would seem at first glance likely to destroy much mica; but I do not believe that this is the case. The miner instantly knows by the feel of his steel when his drill strikes a book of mica. In fact, it is almost impossible to drill through a solid book of mica, for, unless the steel is held perpendicular to the cleavage, the drill cannot be turned. Blasting around a book of mica does not injure it in the least, if ordinary care be used and only enough dynamite be employed to loosen the rock. The Indian miners are patient and skillful in getting out the books with little injury. The large amount of waste and broken mica in and about a mine gives the impression that much good material has been destroyed in mining. But much of the mica in a vein is broken and twisted before it is

mined; and I doubt if an appreciable amount is ruined by the miners. Two miners in mica-bearing ground excavate on an average not more than 2 cu. ft. per day. At such a speed they certainly have time to take all due precaution not to destroy mica. By the adoption of a rational system of stoping instead of working at the bottom of a narrow hole, greater production per man would immediately result, at a much cheaper mining cost; but I do not believe that any more mica of marketable quality would be recovered from the same ground.

In a few cases where the vein was mica-bearing for a width of 7 or 8 ft., open stopes have resulted from the joining together of many meandering holes, forming pillars, which were then attacked, and as far as possible extracted. In most cases of this class, the pillars have been robbed until the coming in of the hanging-wall has made the works inaccessible; and, since mine-maps are practically unknown, the reopening of such places gives rise to many difficult problems. No one knows how deep the mine was; where the bottom workings are; or why mining was stopped,—whether by reason of caving, or too much water, or the poverty of ground. Under these conditions, it is difficult to determine the best location for a shaft, or even whether it is worth while to sink one. According to the old miners, every deep abandoned working was fabulously rich. There is a little reason in what they say, for under old conditions only remarkably rich deposits could be worked to any depth. A view of abandoned workings is shown in Fig. 3.

All drilling is double-handed. Some of the miners can drill an upward as easily as a downward hole. Mucking is done by filling baskets with the hand or with a hoe-like tool called a *kodali*. To move waste even 5 or 6 ft., the baskets are filled, picked up, carried and thrown down.

Although young children and women work in the mines, mining accidents are rare, about 0.53 per 1,000. Most fatal accidents have been due to rock-falls from the sides of open cuts. A system of underhand stoping with pillars supporting a hanging-wall, such as Mr. Podger is adopting in some of the mines of Tata, Sons & Co., using packed walls and waste filling when the dip is not too great, appears to suit local conditions best. In steeply dipping veins, to prove a deposit first by shafts and cross-cuts, and then adopt a system of overhand stoping on

waste filling, would seem to be the best method of development.

In one case I had access to the accounts of a mine over a series of years. The workings were partly accessible, and the amount excavated could be fairly well estimated. This mine was on a vein dipping at an angle of 80° . The stoping width was from 3 to 8 ft. Both the hanging- and foot-walls were in pegmatite, quartz, and unaltered feldspar. The mine is now an abandoned open stope 250 ft. long, supported on small pillars, and mined to a depth of from 60 to 120 ft. The mine had just about paid for itself; and the mica, at the price then prevailing (1904-1907), had brought 1 rupee 4 annas (1s. 8d., or about 41 cents) per cu. ft. of vein material excavated. It has the reputation of being an average mine. Work had been stopped on account of too much water.

Labor.

During the rains little mining is done; not so much because of the extra water as because many of the laborers go back to their fields during the season of cultivation. The labor is chiefly drawn from the neighboring tribes of Hindooized aborigines, a mixed race of low-caste Hindoos, Santalis and Uyrias speaking Hindustani. Wages are from 4 to 6 annas (from 8 to 12 cents) for men, and one-half of that for women and children. Their efficiency is proportional to their pay, if they are allowed to use their own methods; but they are not a bad race physically, and under careful supervision could develop into fairly competent miners. They are a tractable, cheerful folk and give little trouble if carefully handled.

An experiment is being made in importing Pathans, Moham-medans from the northwest frontier, to do contract-work. They earn about 60 cents a day. They are big, strong men and intelligent workers and need little supervision upon a contract. In one case they were driving an adit 6 by 6 ft. in size by hand through fairly hard schist, paying for all supplies except drill steel and blacksmith, for \$3.32 per ft. There is never much trouble with labor, but strikes sometimes occur and the cause is often hard to find, and may seem to a European too trifling to notice when it is discovered (some minute point of caste regulation). There is, however, a method of arbitration not found in Europe or America. Each little village has a Council, or Punchayat, the

head of which is called the Punch, and is elected from among the elders. If a mine-manager tactfully brings up any grievance before the Punch, the trouble is apt to be settled with little ado. In the one instance in which this ancient arbitration board came under my notice, some miners had struck on account of alleged unfair treatment on a contract. The case was presented, argued, and decided against the miners, who went back contentedly to work under the conditions of which they had complained. The ordinary coolie is not intelligent enough to work on contract, except in excavating soft surface ground. He will make no attempt to earn more than his regular pay, but will work shorter hours. Most of the direct supervision is done by Bengali Baboos. Their education has been entirely literary; their ideas of mining are derived from the age-long practices of the aborigines, and to overcome their mental inertia is the hardest task in introducing new methods. Their salaries range from \$4 to \$20 per month.

Preparation for Market.

The mica is roughly sorted by the miners and that which is obviously worthless is thrown on the dump and the rest made up into bundles of about 30 lb., which are brought to the storehouse at the end of the day. For an extra large amount, or some unusually fine piece, the miners receive *bucksheesh*, which is generally given as a stimulus to comparative honesty. The mica is then roughly sorted according to size and given to the trimmers, who are graded according to the size of the material they are to trim. Boys take the small pieces and well-paid men the large mica. The rough edges are cut off with a sickle-like knife, the mica being held against a stake driven into the ground and the knife at an angle of about 45° to the sheet of mica. A group of mica cutters at work is shown in Fig. 5.

All cracks and inclusions are cut away, regardless of what shape the pieces may assume. In the larger sizes each piece is carefully gone over and cleaned as much as possible. This is called sickle-trimmed mica. The best splitters and sorters are highly skilled men and receive the comparatively high pay of from 15 to 32 cents per day. The mica is graded then according to size, as follows:

Sq. In.	Sq. In
Specials,over 36.	No. 4, from 6 to 10.
No. 1, from 24 to 36.	No. 5, from 3 to 6.
No. 2, from 15 to 24.	No. 6, from 2 to 3.
No. 3, from 10 to 15	

The sizes are then regraded according to quality, each manager having his own ideas of what the quality should be. The usual grading is into clear ruby, slightly stained, and black-spotted. The silver-white soft mica is usually graded separately; but the smoky, brown, pale amber and other varieties are mixed in with the ruby. The color of clear mica can only be easily seen in sheets over 0.25 in. in thickness. Stained or spotted mica, No. 6 and some No. 5, and thin sheets derived from cleaning the larger sizes, are split into films. This work is done for the most part by children, occasionally by women. Children are the best workers; older people have not the delicate touch required. The films are split with a small knife or with the thumb-nail, which is allowed to grow long. A good worker can make about 2 lb. a day. They generally do considerably less. They receive from 2 to 3 cents per pound for the splitting. The films should be so fine that a handful of them feels as soft as a bundle of tissue paper. The different sizes and grades are then packed and shipped in wooden boxes holding 100 lb. each. Some mica is cut to size, but this is uncommon in the Hazaribagh district. There is considerable speculation among natives and Europeans in buying mica in small lots, holding it until a large stock has been accumulated, and then selling it. Thus mica is often bought and sold many times before it is finally shipped. Generally in these speculative stocks all sizes and grades are mixed together, and there are many tricks to catch the unwary. It is not a game for the novice to try his hand at.

Mining Laws.

Near Kodarma is a government forest-reserve about 8 miles square. This is divided into 40-acre "squares," which are leased by the government to any responsible person, after rather tedious formalities, at 60 rupees (\$20) per "square." The lease is for 30 years and may be renewed; government sanction must be had to transfer a square. The lessee is required to work continuously on one-fourth of the squares he holds. This

rule, and many of the other regulations which would be vexatious if enforced, are almost ignored. The government forest is the most worked area, and contains hundreds of mines. This is largely due to the fact that the title is good and the terms are reasonable. There is some land which is just as good, outside of the government forest, and all of it has been scratched over; but very little is being done on it now. Outside of the forest, land is held by Indian Zemindars, or landlords, under many different kinds of title from the government. Many of the mines are on undivided estates, constituting a sort of family stock company, each member of the family receiving a percentage from the rent of a given area. The titles are all in confusion with regard to mineral rights. There may be several grades of tenants and sub-tenants as well as the owners on the same land; and who is the rightful owner of the mineral rights can only be determined in each case by lengthy litigation.

Except F. F. Christen & Co., who have acquired Zemindar rights over large areas outside of the government forest, Europeans have done but little mining. The proprietors lease on a heavy nominal royalty of five-eighths of the output; but they have to take the word of the lessee as to what the output is, since there is no way to check his statement. When a mine is leased, it is generally described merely by name and not by boundaries—which has led to considerable trouble. In rare cases only can owners of mineral rights give a clear title, and then it is next to impossible to buy or get a long lease on reasonable terms—in which particular the Indian shows his kinship to many other mineral proprietors.

Throughout this district all the good veins and many poor ones have been attacked, and worked out near the surface. But the surface as a rule has only been scratched over, even in the most developed area of the government forest. At present only about 1 in 40 of the old workings are operating, but there is as much mining going on now as ever. As soon as one place becomes difficult to work it is abandoned for another old working that may be easier; and this, perhaps, will be left in turn, to try the first again. The recent relatively high price of the small sizes has made it more profitable to scratch around the old pits above water-level, getting out what was formerly left

as too small to take trouble with, than to take in hand the more formidable job of opening up the deeper and richer mines. The shallow workings are picked at again and again, bringing down the hanging-wall, and poorer and poorer ground is mined. All the mica easily available by crude methods is nearly gone, a fact which is reflected in the present high prices and small supplies. When it is considered that for many years about 14,000 men, women, and children have been engaged in mica mining, for the main part burrowing for the shallow mica, it is evident that an enormous superficial area has been gone over.

THE NELLORE DISTRICT.

The district of Nellore in the Province of Madras is the only other important mica-bearing area in India, furnishing about one-fourth of the Indian product. There are many points of contrast between the two districts. Instead of the well-exposed hills of Hazaribagh, we have here a low-lying plain covered by recent alluvial deposits. Only an occasional outcrop of the underlying schist and pegmatite can be found. Mica has been mined on a large scale here for only 20 years, and only 35 deposits have been opened. These show lens-like masses of quartz and feldspar, encircled by the folia of inclosing schist. Fig. 4 is a view of the Itakara mine.

The Inikuvti, the largest mine, started on a lens of pegmatite 250 by 150 ft. in size, and dipping at an angle of 60° . It was worked as an open quarry to a depth of about 100 ft. On attempting to start underground mining, insufficient pillars were left; and at the time of my visit, in 1910, only one corner of the mine was accessible. There were only three mines working at that time, and these in only a small way. Steam pumps were used, but all rock was removed by strings of coolies carrying diminutive baskets. Wages are about the same as in Hazaribagh, but the miners are distinctly inferior physically. During the last three years litigation over the most important mines has hindered development. No one has been able to get a title clear enough to warrant sinking capital in development. The best mica here is a light-green variety, of which a large amount of clear sheets was formerly won; but recently only mica of somewhat inferior quality has been shipped.

Preparation for Market.

The mica here, after being roughly trimmed, is cut into the largest possible rectangle with a pair of large garden shears, one handle of which is fixed to a stake in the ground, with the cutting edge vertical. Why it is not shipped in the sickle-dressed condition, like that from Kodarma, I could not understand. There is a decided loss of value in cutting the mica into rectangles, as many uses of mica require irregular shapes, and, furthermore, rectangular mica is considered as "manufactured" under the U. S. tariff rules, and pays 10 cents per pound duty plus 20 per cent. ad valorem, while sickle-dressed pays only 5 cents per pound and 20 per cent. ad valorem. Since the United States is the largest importer of mica, this is a point of material importance in shipping mica to the general market.

Samarskite.

In 1909 samarskite was found in the Sankawa mine, near Nellore, in masses weighing as much as 100 lb. Several tons have been won; but it occurs in isolated masses, which makes prospecting for it difficult.

THE MICA MARKET.

Mincing Lane, London, is the market for nearly all the mica produced in India. Every six weeks there is a sale in London by the produce brokers. Two days before the sale, the brokers have samples of various grades of mica from each shipment on show at a warehouse. These samples are carefully inspected by the mica-buyers, and the shipments are sold at auction on the third day. At these sales Indian mica predominates; smaller lots from South Africa, Australia, Ceylon, Japan, Mexico, Bengal, and Canada are sporadically offered. Most of the mica imported into New York is purchased in London, but mica is sold on contract by large purchasers in India to Germany and the United States direct. On account of the personal factor in grading mica, there can easily be an honest difference of opinion whether a lot is up to standard or not; and to sell by any other method than open auction is a matter of some difficulty. At nearly every sale some utterly worthless mica appears, shipped from some out-of-the-way corner of

the world by some ignorant but hopeful individual, who has heard of the value of mica but does not understand the qualities required to make it marketable. Most of the Indian shipments are made by 10 or 12 firms who mine for themselves and purchase from the small producers. The artificial fluctuation brought about by the speculators and the fluctuating demand for each variety make it difficult for even the best informed to give more than a rough estimate of what a given lot of mica should bring. Giving statistics of the value of mica by the ton without regard to size or quality, as is the custom, may be interesting, as showing the state of the industry, but can give no one an idea of what a particular lot will fetch.

The price of mica in shillings per pound sold at auction in London on Sept. 11, 1912, by Edward Davis & Co. is :

Bengal Ruby.

No.	Clear.		Slightly Stained.		Stained.	
	s.	d.	s.	d.	s.	d.
No. 1.....	8	9	6	6
No. 2.....	7	9	7	3	4	8
No. 3.....	6	0	5	0	4	4
No. 4.....	4	6	3	9	2	10
No. 5.....	2	1	1	3
No. 6.....	0	5 $\frac{1}{4}$
No. 5. Splittings.....	0	7
No. 6. Splittings.....	0	4 $\frac{1}{2}$

In January, 1911, the prices obtained at a sale in same place were :

No. 1.....	8	6	5	4
No. 2.....	5	6	3	6
No. 3.....	4	6	3	0
No. 4.....	3	0	1	6
No. 5.....	1	0	0	7	0 9
No. 6.....	0	5
No. 5. Splittings.....	0	5
No. 6. Splittings.....	0	4

In 1901 the prices given by Sir Thomas Holland (*ibid.*, see p. 93) were :

	Clear.		Stained.		
No. 1.....	5	1	3	6
No. 2.....	3	2	1	9
No. 3.....	1	4	0	10
No. 4.....	0	7	0	5 $\frac{1}{2}$
No. 5.....	0	2 $\frac{1}{2}$

It is interesting to note that condenser-plates, used extensively in wireless stations, and sold at 16 shillings per 100, about 300 weighing a pound, are merely thin sheets of clear, faultless mica 4.5 by 4.25 in. in size. Disks for phonograph sounders, circular pieces of mica about 1 in. in diameter (several hundred weigh hardly an ounce) cost 40 shillings a hundred, but in a lot of clear hard ruby mica only occasional pieces can be found flat enough to use for a diaphragm. A recent order in London for clear, perfect 12 by 12 in. sheets could only be filled at 15 shillings each. Some common forms of prepared mica are shown in Fig. 6.

Because of the friction of trade arising in a small industry, the insignificant amount produced by many small operators, and the many grades and classes of mica coming from even a small producer, it is difficult for the government to collect statistical data possessing even a semblance of accuracy. It is evident to those familiar with the mica market in India that there has been no hoarding of supplies during the last four years; but the government reports that 32,699 hundredweight was produced in 1910, and valued at £54,427; whereas, 39,612 hundredweight, valued at £177,152, was exported. The latter figure represents approximately the average yearly production of India.

During 1900 the United States imported 1,192,221 lb. of mica, valued at \$721,541, of which 10 per cent. was Canadian phlogopite, and the rest largely from India.

The Elk City Mining District, Idaho County, Idaho.

BY ARTHUR L. FLAGG, SPOKANE, WASH.

(New York Meeting, February, 1913.)

THE Elk City mining district of Idaho occupies a position near the geographical center of Idaho county, a region of moderate elevation in the western foot-hills of the Bitter Root mountains. The district comprises an area of approximately 350 sq. miles, bounded on the west by the Newsome district and on the south by the Dixie and Oro Grande districts. There are no mining districts to the north or east.

Elk City, the principal town and the seat of the mining office for the district, is situated in a hydrographical basin surrounded by low hills. The elevation is 4,200 ft. above sea-level. Through this basin flow Elk Creek, Red river, and American river, which unite west of town to form the south fork of the Clearwater. The one township is set out from the Bitter Root Forest Reserve.

Though a railroad starting from the Stites terminus of the Clearwater Short Line of the Northern Pacific railroad has been under consideration for some years, the camp is yet without railroad facilities. The projected road would follow the south fork of the Clearwater on easy grades, but would be very expensive to build and maintain. While such a road would greatly benefit those who are now operating in the camp, and in a slight degree stimulate interest in its possibilities, there is hardly business enough to warrant its construction.

At present all supplies and equipment are hauled in by wagon from Stites, a distance of 58 miles, over a wagon road constructed by the State in 1904, following in the main the old Nez Perce trail. While this is not a bad road, it is not all that might be desired. Starting at Stites at an elevation of 1,327 feet above sea-level, the road runs through low hills to Clearwater, a distance of 10 miles. In the next 16 miles the climb is steady until the Mountain House is reached, which is near the summit of Mount Pisgah. This is the highest point passed on the road, having an elevation of 6,500 ft. From the

Mountain House to Newsome is a descent of 2,300 ft. in the 9 miles, while the next 9 miles is a steady climb to the summit of Elk mountain. From the summit to Elk City, a distance of 9 miles, a drop of 1,650 ft. is made. During five months of the year sleds are used, and, except for the descent from the Mountain House to Newsome, the hauling is good. In the

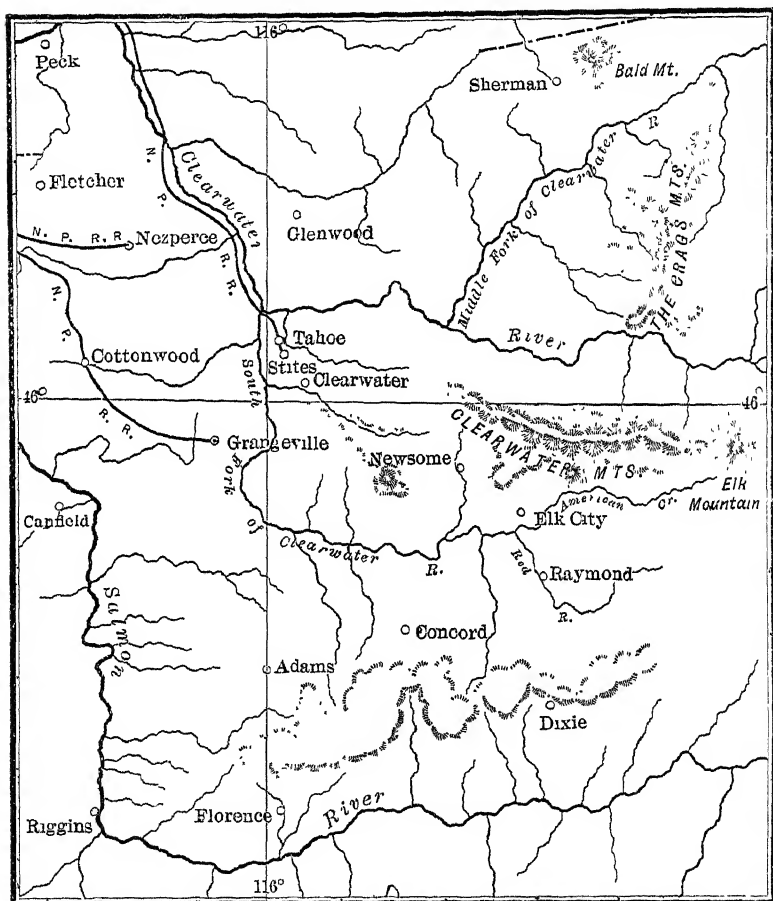


FIG. 1.—MAP OF PART OF IDAHO COUNTY, IDAHO, SHOWING ELK CITY MINING DISTRICT.

early summer after the melting of the snows, the roads are in bad condition owing to the mud. The prevailing freight rate from Stites to Elk City is 2 cents per pound. For shipments out—ore and concentrates—the rate is variable, depending on the number of freighters as well as the season. In general, the rate may be said to average about 1 cent per pound.

There is some agricultural land about Elk City, but the season is so short that no attempt is made at farming on a large scale. There are several good hay ranches, on which a little grain also is raised, though not enough to meet the local demand.

The region is well timbered with tamarack, red and white fir, black pine, spruce, and some yellow pine. Most of the timber being on the Forest Reserve is subject to its regulations. There is not a mining claim which has not a good growth of timber on it suitable for mine uses.

There are good water-power facilities in the district, but at the present time practically none of them are utilized. Almost all of the properties are equipped with steam plants, using wood for fuel. While wood is abundant, local conditions are such that the power-costs are high. The only plants using water-power are the Mascot—a recent installation—and the Last Chance. Power for the Mascot mill is developed by a small Pelton-type wheel running under a 130-ft. head, the water being taken from the old Gold Hill placer ditch.

The history of Elk City begins with the inrush of placer miners following the discoveries of Capt. E. D. Pierce in 1860 on the north fork of the Clearwater. In June, 1861, a group of about 30 men under the leadership of Mose Milner, a noted character in the early history of Idaho, came over into the Elk City basin and camped on Elk creek, near what is now the north end of the town. On June 14 a drawing was made for ten placer claims lying along American river, beginning at a point where the Dixie road crosses this stream and extending down stream. The district was then known as the Union district, for which L. B. Monson was the first recorder. A short time after the arrival of the first party, a second came, under the leadership of Sam Warfield, a famous Indian-fighter. They settled west of town, along Red river and American river, in what was then known as the Central district, which included the Buffalo Hill and Gold Hill placers, both of which were heavy producers later.

On Sept. 1, 1861, Elk City was founded, Captain Monson, the recorder, being made the first justice of the peace. Stores and houses soon sprung up and Elk City was a live mining camp until the Florence excitement, when many of the miners went there in hopes of making richer finds.

In 1872 there was a great influx of Chinese, who bought out the white miners who had not joined the rush to Florence. They dominated the camp until about 1884, when there was a revival of interest in the district which brought back the white miners. It is said that during this period there were as many as 1,500 Chinese in the Elk City district alone. The so-called "skim diggings," which were shallow deposits representing a concentration of placer gold at a depth of 3 ft. or less, covered extensive areas and yielded large returns at the hands of the Chinese. They left little ground unturned that carried gold, often reworking successfully old tailings dumps left by the white miners. Placer mining has now been reduced to properties which must be operated on a large scale by hydraulicking or dredging under conditions involving large outlays of capital for equipment, or high terraces which are beyond the reach of water.

The first quartz location was made by Blaine and Hancock, who staked what is now known as the Buster mine. The exact date of their location is not known, since it was never recorded. However, they did no work on the property and it was relocated about 1870 by Samuel W. (Buster) Smith and H. E. McKinney, who recorded it Aug. 20, 1884, it appearing then as the first quartz location recorded in the district.

The first attempt to mill any of the ores was made by "Buster" Smith in 1870, who built and operated an arrastre to treat the ores from the Buster mine. The first machinery installed was on what is now known as the Congress claim, then called the Red Stone. A Chilean mill was set up but never operated. Later Smith put up a one-stamp mill on his claim, but it was never used. The first complete plant was the ten-stamp mill erected on the American Eagle property in 1902.

No entirely satisfactory figures can be obtained regarding the gold production of Elk City district since its discovery, though it is generally believed to be somewhere in the neighborhood of \$10,000,000,¹ of which the greater part was derived from placer operations.

The Elk City district, lying in the western foot-hills of the rugged Bitter Root mountains, is a region of low hills of moderate elevation, perhaps in part the remnant of an old plateau

¹ *Report of the Commissioner of Immigration*, p. 119 (1905).

of which the original features are more or less obscured by the complicated system of ridges and cañons. For the most part the surface is covered to considerable depth with drift, composed chiefly of interstratified clays and gravels with few larger boulders. The soil covering is moderately deep, subject to wide variation. In some of the placer diggings, notably at Buffalo hill, pot-holes are found along the axis of old river channels in the bed-rock, and cobbles of considerable size are not infrequently found imbedded in the decomposing gneiss, a condition to be attributed to the weight of superimposed gravels rather than the direct evidence of glacial action, as has been suggested.

The formation of the district is almost entirely granite and schistose gneiss, together with more highly metamorphic schists which perhaps are of sedimentary origin. All transition phases of granitic rocks are often seen close together in small areas. The prevailing rock is a light-colored biotite-gneiss which is evidently derived from a granite. Excessive dynamo-metamorphic stresses have developed fine-grained, dark-colored phases tending towards schists. The dip as a rule is low, but subject to wide variations. The gneiss is often highly contorted and crumpled perpendicularly to the strike. The essential constituents of the gneiss are orthoclase, oligoclase (plagioclase rarely), abundant quartz, and varying quantities of biotite and muscovite. Lenticular crystals of orthoclase surrounded by narrow, dark rims of biotite have been developed, giving the gneiss an ocellar structure.

Everywhere the gneiss is cut by dikes of various widths. While there is a wide difference in the structure among these dikes, they seem to be essentially of the same composition. Narrow aplite dikes cut the gneiss parallel to the schistosity, while others, identical in form and composition, cross the formation. Large dikes of pegmatite, often greatly sheared, are also found. In some areas where excessive compression has developed a fine-grained, highly schistose structure characterized by sinuous bands of biotite and muscovite, an incipient gneissoid structure is noted in the dikes. The bed-rock of the Buffalo Hill placer is cut in all directions by moderate-sized dikes of a medium-grained granite, some of which are more or less sheared.

I noted one diorite dike of moderate width, exposed in the

bed of Red river about 2 miles up stream from its junction with American river. Others have been reported from the district, and several are known to occur in the Buffalo Hump region, about 26 miles south.

The veins of the district are usually large and well defined, cutting across the schistosity of the formation in a general E-W. direction, and with few exceptions stand at high angles, from 60° to 70° from the horizontal, dipping to the south. As a rule, they are fairly regular in dip and strike. The only faulting so far encountered has proved inconsequential, all faults being normal with but slight displacement.

The major part of the vein filling is quartz, though rarely is the entire mass of material found between walls of commercial value. A cross-section of a 4-ft. vein near the center of the district may be taken as typical of the prevailing vein structure in the camp. Next the foot-wall is found 6 in. of talc and clay gouge, fairly regular and representing the course of meteoric waters. A streak of quartz, considerably shattered, about 12 in. wide comes next. This quartz carries good value, and is separated from the main body of quartz by a mass of decomposed granite, 8 in. wide, which varies in width in different parts of the vein. The nature of this barren rock, as well as its mode of occurrence, lead to the supposition that it represents a portion of the country rock detached from the foot-wall side and surrounded by vein matter during the process of vein filling. The main body of quartz, 27 in. wide, somewhat firmer than the narrower streak next the foot-wall, lies between the granite inlier and the hanging-wall, being separated from the latter by a narrow streak of talc.

One very interesting feature in connection with the vein formation which will warrant more study is the frequent occurrence of a narrow dike of fine-grained biotite-granite on the foot-wall side of the vein. These dikes, which range in width from a few inches to 2 ft., seem to be identical in composition with the numerous dikes mentioned above, but differ in structure. Those found in conjunction with the veins are uniformly even and moderately fine-grained. In some instances an incipient gneissoid structure, similar to that observed in connection with other dikes, is developed. This is due to a readjustment of the micas in more or less parallel

lines, together with compression and elongation of the other constituents. The close association of these dikes and veins has suggested the hypothesis of a granite filling for the original fissures, which were opened subsequently to receive the quartz carrying the gold, silver, and other metals.

The ores so far developed in the camp are all highly oxidized, carrying gold in small grains and flakes. Not infrequently, at no great depth from the surface large nodules of unaltered sulphides are found imbedded in the quartz, occurring as a rule on the hanging-wall side of the vein. Sulphides, more finely disseminated, occur in variable quantities in almost all the ores. In some of the specimens showing unaltered sulphides small flakes of native gold have been observed on the faces of the pyrite crystals. The firmer quartz frequently shows fine-grained galena and tetrahedrite, though these minerals are not abundant. Some tellurides have been reported from the district, but no positive evidences have come under my observation.

Though Elk City is primarily a gold district and all the prospecting has been done for gold, it is not improbable that some silver ores of commercial importance will be developed. The most promising indication is a wide vein on the Revenue property, formerly known as the California. The vein extends from a point south of Gold Hill placer on the east bank of Red river, eastward to the American river. Openings have been made at several places along the vein, the most important of which are tunnels driven in opposite directions from the two rivers named. This vein has the same dip and strike common to the district, though it varies somewhat in structure. The vein filling is a firm white quartz carrying some argentite and galena with variable quantities of stibnite. At the western extremity of the vein the stibnite predominates; the silver values are low and gold is a negligible quantity. In the tunnel running from American river the silver values are higher, the gold values increase materially, and there is less antimony. The vein, which varies greatly in width, seems to have been formed on the foot-wall of a highly-sheared, medium-grained granite dike. In the most westerly tunnel the sheared condition of the granite dike is most plainly seen, and here the granite also is more or less impregnated with stibnite in microscopic crystals. The presence of antimony makes this ore undesirable, though there are some

indications of high-grade silver ores low in antimony on the eastern half of the vein which may prove to be of commercial importance.

One other occurrence of stibnite was noted in Deadwood cañon. With the stibnite is associated a small amount of galena, and the ore is said to run high in silver. While interesting from a scientific point of view, it is probable that the occurrence will never be of commercial importance. Monzonite sands were almost always found in connection with the placer gold and other metals more or less rare are reported from the district, but at best they are in small deposits of little or no importance.

Owing to the fact that as yet no extensive milling operations have been conducted in the district, it is impossible to go into any detailed discussion of reduction processes. From the occurrence in place it seems probable that the ore cannot be broken in the mine in a condition suitable for milling without a coarse sorting, while operations on a large scale may prove the value of a closer sorting of the mine product before it is put into the mill bins. However that may be, there is little doubt about the next step in the treatment of the ore. All the ores in the district so far discovered are amenable to amalgamation. The saving on the plates varies from 25 to 75 per cent., the pulp issuing from the battery at from 20- to 40-mesh size. Concentration should probably follow in all cases, as considerable percentage of the initial values are locked up in the sulphides. In most instances the tailings from the tables are not of sufficient value to justify cyanidation. So far as known, no extensive experiments have been tried on the concentrates to determine the possibilities of treating them by cyanide, though it is generally believed that they will require preliminary roasting. With the development of the camp the details of metallurgical practice will be worked out more conclusively, and in my opinion a process will be devised for the complete recovery of all the values on the ground, obviating the necessity of shipping concentrates.

Since the revival of interest in the camp in 1884 there has been considerable prospecting and many promising veins have been opened up to some extent. However, no mining has been done on a large scale in any property. The greatest amount of development work has been done at the South Fork, Buster, and American Eagle mines. The district has had a record

similar to that of many others, having suffered its share of the ills attendant on "wild-catting." The landscape is marred here and there by idle mills and abandoned camps which bear mute but strong testimony to the undesirable side of mining.

There are five mills operating in the district. The Elk City Mines Corporation, of Spokane, owning the South Fork mine, about 7 miles west of town, has operated a five-stamp mill for the past two years. Early in the present year a James table was installed and a very satisfactory recovery is being made, though the mill is not up to the capacity of the mine. A new 10-stamp mill will be built early this year, to be operated by hydro-electric power derived from the south fork of the Clearwater. After taking all things into consideration, this property has without doubt had the best record of any in the district. When the property was first located high-grade ore was encountered in doing the location work, and two carloads, averaging \$200 to the ton, were shipped that year. The locator, E. E. Espy, owned a two-stamp mill which he moved from an adjoining property to the site of the present mill. He treated the ores which were too low grade to ship and, though his mill could handle but 1 ton a day, he made a small fortune out of the property while he operated it. The present plant was erected about four years ago and may almost be said to have operated continuously and successfully ever since.

The Buster mill, built in 1907, and which treated successfully the ores of the Buster mine before it was closed down, was overhauled recently and is treating the ores from the Mineral Zone mine. The equipment consists of 10 stamps, plates, 6 vanners, and a 40-ton cyanide plant. When operating on the Buster ores a total extraction of approximately 90 per cent. was made, 40 of which was recovered on the plates. The tailings from the vanners, after being classified, were treated by straight lixiviation. The slimes required agitation.

Southwest of town a one-stamp mill has operated all summer on the Little Butte property. The Last Chance mill, consisting of three stamps, operates spasmodically with good results. The Mascot mill, east of town, is handling about 20 tons daily in a 3.5-ft. Huntington mill.

It is expected that the American Eagle mill, 7 miles east of town, will soon be in operation again. This was the first com-

plete plant erected in the district. Work has been resumed on the mine, and the mill is being put in running order to be ready for operation as soon as ore is available. During the first year this mill produced \$75,000 in bullion, and practically exhausted the ore which had been developed. The management had not taken the precaution to open up new ore reserves, since it involved sinking. When the ore above the tunnel level was stoped out the property was closed down and lay idle several years. Recently miners working under a lease opened up high-grade ore in a winze 30 ft. below the tunnel-level, but they were unable to handle the water. The Rio Tinto Mining Co., of Ontario, then secured an option on the property and began operations. Their plan of development provides for sinking the winze 500 ft. below the tunnel level and driving the necessary drifts.

In the center of town is a small custom plant erected recently by public subscription, equipped with three stamps, plates, concentrating tables, Garver gravity process roaster for roasting concentrates, and a cyanide plant. It was expected that this plant would treat small lots of high-grade ore, of which there is no small quantity to be found in the various prospects about the district, and thus stimulate interest in the camp, besides furnishing the prospector and small operator of limited means a method by which they might realize a little ready money for the further development of their claims. Owing to unsatisfactory management the mill has not been operated extensively, and so far has failed of its purpose.

At the present time there is more or less activity in the district, though it lacks the snap and vigor of a live mining camp. It is hampered to a great extent by the accumulated prejudice resulting from unsuccessful booms, as well as by a few striking examples of mismanagement; and to a much lesser extent by the lack of railroad facilities. Those who are operating in the district at present are, for the most part, men of limited means but a commendable amount of pluck and energy. There can be but little doubt about the future of the camp if the much-needed development work opens up ore in quantity and quality commensurate with what has already been disclosed in the present shallow workings.

Fire-Clay Deposits of Canada.*

BY HEINRICH RIES, ITHACA, N. Y.

(New York Meeting, February, 1913)

WITH the rapid development of Canadian industries calling for the use of fire-brick for the lining of furnaces or cement-kilns, for constructing brick-kilns or coke-ovens, for lead-furnaces, etc., the question has often arisen whether the provinces within the Dominion have not the necessary raw materials for the manufacture of such refractory wares.

As an indication for the need of such products in Canada, we find that the imports in 1910 amounted in value to \$519,454. The materials came in part from Great Britain, and part from the United States. Those supplied by the latter country are obtained largely from Pennsylvania and Ohio, and when shipped to distant points, such as British Columbia, the item of transportation is a serious factor.

Scotch fire-brick is used both in eastern and western Canada, but in the latter case the shipments are made by water route to Vancouver.

During the past four years I have been investigating the clay-deposits of Canada, for the Canadian Geological Survey, and while all types of clay have been included in the study, special pains have been taken to ascertain the occurrence of refractory clays. As a result of this work we are able to report on the existence of beds of refractory clays at the following localities: Inverness and Shubenacadie, Nova Scotia; St. Remi, d'Amherst, Quebec; Dirt Hills and Cypress Hills, Saskatchewan; Sumas Mountain, near Clayburn, B. C., and Kyuquot, on northwestern Vancouver Island. A map of Canada, showing the fire-clay deposits, is given in Fig. 1.

It is possible that other deposits may be discovered in the future in districts remotely located from the present lines of railroad.

* Published by permission of the Director of the Canadian Geological Survey.

Of the occurrences mentioned only those of Sumas mountain and Kyuquot are at present utilized. The vast deposits of the Dirt Hills are still attractive for development.

The St. Remi deposit is not discussed in this paper, since it is of more value for making paper than for fire-brick.

A fact which enhances the value of some of these deposits is that they are adapted not alone to the manufacture of fire-brick, but can in some cases also be worked for use, wholly or in part, in the production of stoneware, terra cotta and sewer-pipe.

I. NOVA SCOTIA.

A most remarkable clay-deposit of undetermined age is found in the Musquodoboit valley, and near Shubenacadie, both areas being a short distance north of Halifax, the latter along the line of the Intercolonial railway. The material is a highly-plastic clay, of dark gray, white, or mottled red and white color, lying beneath the glacial drift, and resting probably on bed-rock. Scattered lumps of lignite were found in the clays of both localities, and it was hoped that thin sections of it might aid in determining the age of these beds, but the material is not sufficiently well preserved to be of value in this respect. It seems probable, however, that these clays may be of the same age as the Cretaceous fire-clays of New Jersey, since they bear a remarkably strong resemblance to them.

It is very difficult to outline the exact area underlain by this deposit, owing to the mantle of glacial drift overlying the region, but the fact that the material is found at several points, extending over a distance of several miles, indicates its possible extent. A number of borings were made at those points where the drift cover was thin.

1. *Musquodoboit Valley*.—The broad valley of the Musquodoboit river is bordered by rather high, even ridges with gentle slopes, while the valley is floored with glacial drift, which overlies unconsolidated sediments of unknown depth, and referred to as underclay. This underclay outcrops at a few points along the banks of the river at the village of Middle Musquodoboit, and for several miles above, also on Murphy brook and Paint brook, two small streams entering from the north.

The underclay has attracted attention for many years by

reason of its bright color and high plasticity. But at most points it is covered by a stiff boulder clay, in which lumps and patches of the underclay are often included.

A limited number of borings made revealed the fact that the deposit is not one solid mass of high-grade clay, but that it is a stratified formation made up of fine white clay, colored clay, and sands, with occasional layers of lignite, and some scattered pyrite concretions. In other words it closely resembles many of the New Jersey Cretaceous clay-occurrences. Many of the colored portions bleach strongly on exposure to the weather.

A number of borings, made at different points,¹ indicated thicknesses of at least from 25 to 50 feet.

The plasticity of the clay is usually good, even the silty beds possessing fair plasticity—in fact, enough to permit their being molded.

Along the flood plain of the river there are some large patches of very stiff and very adhesive clay of light gray or bluish color, which the local residents regard as fire-clay, but it is not.

The deposit in this valley is undoubtedly extensive, but before developing it a number of additional borings should be made, as in the Geological Survey work it was not possible to spend the time and money on it that a private corporation could.

The physical tests given in Table I. represent the properties of, Sample I., a red and gray mottled clay from Murphy's brook, and Sample II., a light gray clay from farther up the same brook.

TABLE I.—*Physical Tests of Clays from Murphy's Brook.*

	Sample I.	Sample II.
Quantity passing 200-mesh sieve, per cent	91.6	99
Water required for mixing, per cent.....	30.8	
Air-shrinkage, per cent.....	6.5	6.8
Average tensile strength, lb per sq. in.	68	81
Cone 010.		
Fire-shrinkage, per cent	0.1	0.4
Absorption, per cent.....	21.68	19.3
Color.....	Pink.	White.

¹ For details of these bore-holes see *Canadian Geological Survey Memoir* 16 E, p. 77 (1911).

Cone 05.

Fire-shrinkage, per cent.	2.6	2.3
Absorption, per cent.	18.29	16.71
Color.	Salmon.	White.

Cone 03.

Fire-shrinkage, per cent.	6.0	6.0
Absorption, per cent.	12.96	15.92
Color.	Salmon.	White.

Cone 1.

Fire-shrinkage, per cent.	6.3	6.0
Absorption, per cent.	7.0	7.41
Color.	Pink.	White.

Cone 3.

Fire-shrinkage, per cent.	6.3	7.0
Absorption, per cent.	5.41	7.71
Color.	Pink.	Cream.

Cone 5.

Fire-shrinkage, per cent.	7.3	7.3
Absorption, per cent.	3.66	4.89
Color.	Red.	Cream.

Cone 9.

Fire-shrinkage, per cent.	9.0	8.0
Absorption, per cent.	0.29	4.84
Color.	Red brown.	Cream.

Fusion-point, cone.	27
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Sample I. burned steel hard at Cone 03; its form was still preserved at Cone 9, but the fire-shrinkage was somewhat high. It is not a good fire-clay.

Sample II., which can be regarded as a fire-clay, did not become steel hard until Cone 1, due to its more refractory character. In its natural condition it burns rather dense for a fire-brick, but the body could be opened up by the addition of ground-up brick.

The chemical analyses of the clays are given in Table II.

TABLE II.—*Composition of Clays from Murphy's Brook.*

	I. Per Cent.	II. Per Cent.	III. Per Cent.	IV. Per Cent.	V. Per Cent.
Silica	55.14	53.20	63.91	52.90	53.00
Alumina.	28.84	30.25	18.60	29.00	32.10
Ferric oxide.....	1.91	1.72	5.75	3.20	1.70
Titanic oxide.....	2.37	1.47	undet.		undet.
Magnesia.	0.25	trace.	trace.		trace.
Lime.	0.38	none.	trace.	undet.	none.
Soda.	0.48		undet.		
Potash.	1.88	1.33			{ 0.97
Loss on ignition..	9.24 ^a	12.00	10.30	12.10	
	100.49				

^a Chemically combined water.

I. Gray clay, the physical tests of which are given in Table I., column II. M. F. Connor, Dept. Mines, analyst.

II. Average sample of white clay from 18-ft. bore-hole.

III. Average sample of mottled clay from 15-ft. bore-hole.

IV. Floated mottled clay.

V. Floated white clay.

Analyses II. to V., by F. H. Mason, Nova Scotia Mining Institute, 1901.

2. *Shubenacadie*.—A light gray plastic clay is exposed along the Intercolonial railway tracks about 0.75 mile south of Shubenacadie station. Small quantities were mined a few years ago, and shipped to a pottery near Enfield on the same line. More recently an effort was made to place the clay on the market, and a shaft was sunk by E. Thompson on his property near the original exposure. This shaft is about 30 ft. deep, and penetrates marine- and boulder-clay for 20 ft., and gray clay for 10 ft. About 7 ft. of the latter is exposed in the face of three short drifts leading from the shaft, and a boring made in the bottom of the latter showed an additional depth of 16 ft. of silty gray clay.

Borings made around Shubenacadie have shown the presence of the clay at a number of points, but sometimes the overburden is considerable, so that the winning of the clay would have to be done in part by underground methods.²

Table III. gives, I., the tests of the light clay from the shaft at Shubenacadie, and II., tests of a carload-lot taken from the shaft, and shipped to the brick-works at Westville, near New Glasgow.

TABLE III.—*Physical Tests of Clays from Shubenacadie.*

	I.	II.
Quantity passing 200-mesh sieve, per cent.....	74
Water required for mixing, per cent.....	22
Air-shrinkage, per cent.....	5.8	6.3
Average tensile strength (lb. per sq. in.).....	110
Cone 010.		
Fire-shrinkage, per cent.....	0.6	0.
Absorption, per cent.....	14.23	14.9
Color.....	White.	White.

² Details of the borings will be found in *Memoir 16 E., Canadian Geological Survey*, p. 84 (1911).

Cone 05.

Fire-shrinkage, per cent.	0.6	0.6
Absorption, per cent.....	13.46	13.9
Color	White.	White.

Cone 03.

Fire-shrinkage, per cent.....	2.0	1.6
Absorption, per cent.....	11 0	12.05
Color.....	White.	White.

Cone 1.

Fire-shrinkage, per cent.....	2.0	2.0
Absorption, per cent.....	10.74	9.95
Color.....	White.	Cream.

Cone 3.

Fire-shrinkage, per cent.....	2.0	3 0
Absorption, per cent.....	10 64	9.1
Color.....	White.	Buff.

Cone 5.

Fire-shrinkage, per cent.....	2.0	3.0
Absorption, per cent.....	10.04	9.0
Color	Cream white.	Buff.

Cone 9.

Fire-shrinkage, per cent.....	4.0
Absorption, per cent.....	4.41
Color.....	Spotted buff.
Fusion-point, cone.....	30.

Sample I. is nearly steel hard at Cone 03. It is a fire-clay, but could be also used in the manufacture of pressed brick and terra cotta.

Sample II. gives a good hard body at Cone 03. The fire-shrinkage is low, but the clay is somewhat dense burning, and would have to be opened up with grog.

An analysis of the clay made by M. F. Connor, of the Department of Mines, Ottawa, gave:

	Per Cent
Silica.....	74.03
Alumina	17.30
Ferric oxide.....	1.15
Titanic oxide.....	1.04
Magnesia	0.16
Lime.....	0.38
Soda	0.53
Potash.....	0.88
Water.....	4.78
	<hr/> 100.25

3. *Inverness*.—There are so many coal-bearing areas in which fire-clays are found under coal, that it has become a wide-spread custom among coal-miners to call every underclay a fire-clay. Nova Scotia is no exception to this rule; but, although there are several coal-areas within the province, there is only one known occurrence of refractory clay, and this is at Inverness.

There, overlying the Hussey seam, is a bed of clean-looking, plastic, gray clay which varies from 18 in. to 3 ft. in thickness. The clay is free from coarse sand, but occasional nodules of pyrite are found. There is not a little fine sand, however, for only 58.4 per cent. of the clay passed through a 200-mesh sieve. The clay worked up with 30.8 per cent. of water, had an air-shrinkage of 8.5 per cent., and an average tensile strength when air-dried of 206 lb. per sq. inch.

Burning-tests on this clay gave:

Cone.	Fire-Shrinkage. Per Cent.	Absorption. Per Cent.	Color.
010	0.3	15.74	Pink buff.
05	4.3	9.79	Buff.
03	2.25	Dark buff.
1	6.6	Buff.
3	7.3	0	Drab.
5	10.0	0	Drab.
9	5	0	Gray.
25	Fused.		

This clay can be classed as one of moderate refractoriness. In its natural form it is to be properly classed as a stoneware clay, but if "grogged" it can be used for the lower grades of fire-brick. If worked, it would have to be done in conjunction with the mining of the underlying 13-ft. seam.

An analysis, made by M. F. Connor, of the Department of Mines, Ottawa, gave:

Silica.....	55.52
Alumina.....	26.80
Ferric oxide.....	2.58
Titanic oxide.....	1.50
Magnesia.....	1.05
Lime.....	0.25
Soda.....	0.73
Potash.....	3.43
Water.....	8.89

100.25

II. SASKATCHEWAN.

The fire-clays of Saskatchewan province all occur, so far as is known, in the Laramie formation, which underlies a large triangular area in southern Saskatchewan. The base of this triangle forms the southern boundary of the province, as far

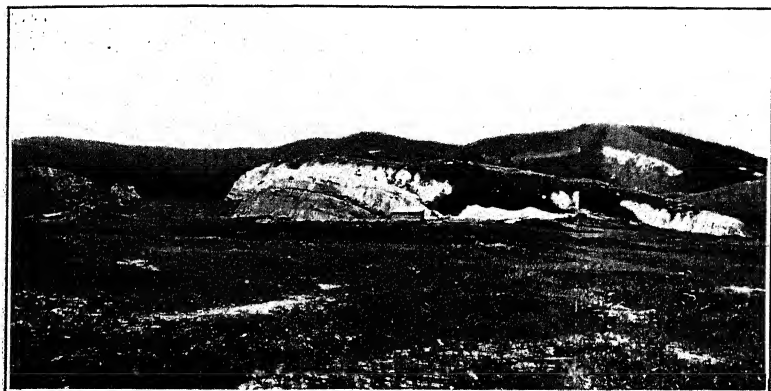


FIG. 2.—VIEW LOOKING SOUTH FROM MARGIN OF DIRT HILLS, SASKATCHEWAN, SHOWING OUTCROP OF WHITE AND GRAY FIRE-CLAYS.

The outcrop on extreme left is a lower-grade clay which can be mixed with the fire-clays to make sewer-pipe.

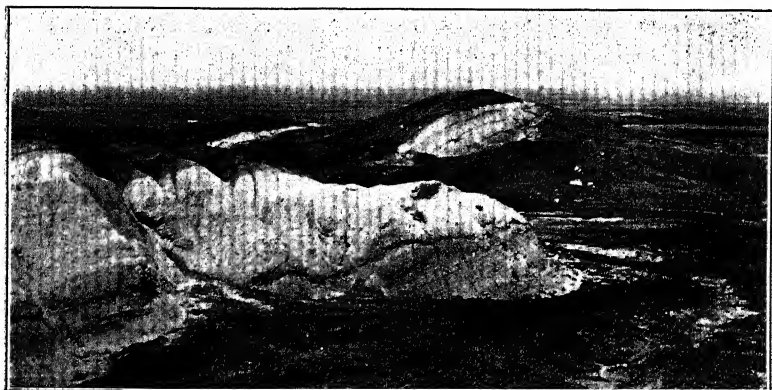


FIG. 3.—OUTCROPS OF WHITE FIRE-CLAY, DIRT-HILLS, SASKATCHEWAN.

west as the Wood Mountain district, which is included in it. From the apex of the triangle a narrow belt extends NW. to a little beyond the main line of the Canadian Pacific railway,

west of Moosejaw. This area, it will be seen by reference to a map, includes the Souris coal-field and the Dirt hills.

Detached areas are found west of this, where the Laramie formation occupies the summits of some of the plateaus, and portions of elevations such as the Cypress hills.

The Souris coal-field contains no fire-clays so far as could be ascertained. The Cypress hills has white-clay deposits, probably of refractory character, but they lie too far from the railway lines to be available.

The Dirt hills area has extensive deposits of refractory character.

1. *Dirt Hills*.—These hills form an isolated elevation rising from the plains about 30 miles south of Moosejaw. A smaller group lying on their northwestern side is known as the Cactus hills. Two lines of railroad passing close to them are almost completed. One of these, the Canadian Northern, is on the east, and the other, the Moosejaw-Lethbridge branch of the Canadian Pacific, is on the NW. boundary.

While the clay-deposits in the hills are undoubtedly of considerable extent, the best natural exposures are found on the north side of the mass, where landslide masses afford excellent exposures. This is in section 28, township 12, range 24, west of the 2d Meridian.

The general topography of these landslide masses is well shown in Figs. 2 and 3, which might lead one to infer that the clays are all white, but while these predominate, other types are not wanting. The series includes white and grayish-white sandy clays, bluish and purplish clays, brown siliceous clay shales, and gypsiferous shales.

The following condensed statement will serve to show the character of the clays in this locality.

At the eastern edge of the group of knolls is a series of alternating red and brown siliceous shales, which have been referred to by some as sewer-pipe clays, but which alone cannot be used for that purpose. The individual beds of this series differ somewhat in their sandiness, and if used the entire series should be mixed together. A few scattered sandstone layers are present, but these are soft and could be easily crushed up.

Overlying these siliceous clays, and separating them from

the next knoll on the west is a series of soft sandstone beds containing large scattered concretions. These are valueless.

Above these stratigraphically, but lying at the base of the second knoll going westward, are heavy beds of gray and grayish-white sandy clay (No. III.), and brownish-gray clay (No. II.), the two being separated by a thin layer of lignite. The brownish-gray clay forms a bed about 20 ft. thick in the lower part of the knoll.

The grayish-white clay (No. III.) contains lenses of smooth white clay (No. IV.). In most cases it would not pay to separate this white clay in mining, unless the lenses are larger.

Table IV. presents tests of these four types of clays.

TABLE IV.—*Physical Tests of Clays from Dirt Hills.*

	I.	II.	III.	IV
Water required, per cent....	30	27	30
Air-shrinkage, per cent....	3.3	8.5	6.1	7.7
Lb. per sq. in.	334	123
Cone 010.				
Fire-shrinkage, per cent....	1.7	0.2	0	0.35
Absorption, per cent.....	9.7	16.38	18.58	16.74
Color.....	Light red.	Pale red.	Whitish.	Cream white.
Cone 03.				
Fire-shrinkage, per cent....	9.3	4.4	2.7	3.7
Absorption, per cent.....	5.5	7.52	15.41	10.34
Color ..	Red.	Light red.	Whitish.	Cream white
Cone 1.				
Fire-shrinkage, per cent....	1.33	5.4	5.3
Absorption, per cent.....	0	4.78	7.67
Color.....	Brown.	Brown.	Cream white.
Cone 5.				
Fire-shrinkage, per cent....	2.7	6.6
Absorption, per cent.....	10.70	4.67
Color.....	Whitish.	Cream white.
Cone 9.				
Fire-shrinkage, per cent....	3.3	6.6
Absorption, per cent.....	9.81	2.60
Color.....	Whitish.	Cream white.
Cone of fusion.....	32	31

Sample I. cracks badly, unless preheated at 500° C., and the tests were made on the preheated clay. It has too high a fire-shrinkage, but works all right if mixed with some of the other clays in the section.

Sample II. has to be dried slowly to avoid cracking, and is not a good fire-clay.

Sample III. stands rapid drying, and is a good fire-clay.

Sample IV. is likewise refractory, but shows higher shrinkage, and denser burning qualities, due to removal of fine sand by washing.

The white clays are also found to the southward in Sec. 24, township 12, range 24, west of the 2d Meridian, where some lignite-mining has been done, but the clay is not so well exposed, and the actual thickness is not known; but the tests of it are interesting as showing the persistence of the refractory beds. They need not be detailed here; suffice it to say that the clay has a fusion-point equal to that of Cone 31. If this white clay is washed, it yields a product free from grit, and which closely resembles the ball-clay used in the manufacture of white-ware.

For purposes of further comparison Table V. gives the composition of clays from several localities. The clay from Dickinson belongs to the same formation as the Dirt Hills clay:

TABLE V.—*Composition of Various Clays.*

	I.	II.	III.	IV.
Silica	51.94	51.92	51.56	66.55
Alumina.....	33.62	32.80	35.04	23.22
Ferric oxide ..	1.5	1.13	0.78	1.16
Lime.....	0.23	0.03	tr.	0.29
Magnesia.....	trace	0.44	tr.	0.61
Potash.....	0.82	0.40	tr.	undet.
Soda.....	0.22			
Loss on ignition.....	11.44	13.49	12.50	7.09

NOTE.—I, washed Dirt Hill clay; II, Salina, Pa.; III, Woodbridge, N. J., and IV, Dickinson, N. D.

Associated with the fire-clays at Dirt hills is also a stoneware clay which fuses at Cone 23. But of more interest is the fact that a mixture of the fire-clay, the stoneware clay and a third clay from this locality makes an excellent sewer-pipe body which takes a good salt-glaze.

2. *Other Localities.*—White clays are reported by D. B. Dowling, of the Canadian Geological Survey, from Wood mountain, and others have been noted by R. G. McConnell, as occurring in quantity on the Frenchman river in southwestern Saskatchewan.

A sample collected from the summit of the Cypress hills

III. BRITISH COLUMBIA.

1. *Sumas Mountain*.—One of the most interesting series of clay-deposits is that lying in Sumas mountain, east of Clayburn, on the Seattle branch of the Canadian Pacific railroad. These deposits consist of a series of shales, sandstones and a few conglomerates, and coaly beds, the whole series having a gentle southwesterly dip.

Near the base of the series there are three beds of distinctly refractory character, the section involving:

	Feet.
Sandstone.....
Upper fire-clay.....	8
Coal, with flint-clay partings.....	0.5 to 1
Lower fire-clay.....	16
Ferruginous clay.....	4
"China-clay".....	from 10 to 15

The flint-clay, as tested from a sample taken in the mine of the Clayburn Clay Co., is a feebly plastic material, with an air-shrinkage of 3.5 per cent., and when fired at Cone 9 having a fire-shrinkage of 5.7 per cent., and an absorption of 7.1 per cent. It fused at Cone 33.

The lower fire-clay is a hard, dark gray material, which worked up with 15 per cent. of water to a mass of moderate plasticity having an air-shrinkage of 3 per cent., and an average tensile strength of 65 lb. per sq. inch.

Firing-tests gave:

Cone.	Fire-shrinkage. Per Cent.	Absorption. Per Cent.	Color.
1	1.3	9.3	Cream.
3	2.0	9.7	Cream.
5	2.0	10.0	Buff.
9	3.0	8.5	Buff.
30	Fused.		

This clay shows low fire-shrinkage, moderate absorption, and burns to a good fire-brick body. Other samples tested showed a fusing-point of Cone 32. A sample tested from the deposits of the Kilgard Co. on the other side of the mountain gave similar results. The product competes successfully with Scotch and Pennsylvania brick.

The so-called china-clay, which underlies the lower fire-clay, is a fine-grained, whitish shale, sometimes smooth and soft, at other times hard with a conchoidal fracture. Throughout this

clay are scattered numerous small limonite spots. It is not a fire-clay as it fuses at Cone 22-25, according to samples tested. Careful grinding and washing gave about 18 per cent. of washed product, which was quite plastic, and worked up into a smooth, but not perfectly white-burning body. It cannot be used for china, but could be employed as an ingredient of a cheaper pottery or wall-tile body if necessary.

2. *Kyuquot, Vancouver Island*.—On the western side of Vancouver Island, near its northern end, there is a deposit of residual clay derived partly from a metamorphosed volcanic rock, presumably a rhyolite. The product as shipped consists of a mixture of clay and lumps of the partly-decomposed rock, and is of distinctly refractory character, since its fusion-point is Cone 30. Its occurrence is interesting, because few residual fire-clays are worked in North America, and because of its occurrence in a glaciated region. Moreover, it is sufficiently extensive to be of commercial value.

CONCLUSIONS.

From what has been said above it will be seen that there are several possible sources of supply of refractory clays, some of which are sufficiently extensive to support a healthy industry. With the rapid industrial development of Canada, there is an increasing demand for refractory bricks of all sorts, and it is safe to predict that many of the deposits now lying idle will be put to a good use in the near future.

The Geographical Distribution of Mining Development in the United States.*

BY EDWARD W. PARKER,† WASHINGTON, D. C.

(New York Meeting, February, 1913)

AT the Cleveland meeting of the Institute, October, 1912, I had occasion to call attention to the general though erroneous impression that the principal mining activities of the United States lie west of the 100th meridian of longitude. Few, I believe, realize that the States east of the Mississippi river, which comprise only 28.8 per cent. of the total area of the United States, exclusive of Alaska, contribute about two-thirds of the total mineral production, computing by values. In 1910 the percentage of the production credited to the Eastern States was 67; in 1911 it was 65. When I have, with becoming modesty, ventured to speak of these comparisons, I have usually been met with the rejoinder, "But you include coal." I surely do, for whereas coal (as petroleum, as natural gas, as limestone) is not in the strictly technical definition of the word, a "mineral," it is legally mineral, is certainly a mineral substance, and contributes by far the largest tonnage and approximately one-third of the total value of all mineral products. The production of coal-mines in 1911 weighed approximately 500,000,000 tons. In 1912 it exceeded that record by nearly, if not quite, 50,000,000 tons. According to H. D. McCaskey, of the U. S. Geological Survey, the total weight of all the metallic ores mined in the United States in 1911 was 105,000,000 tons (a little more than one-fifth that of coal). In 1910 Mr. Lindgren estimated the total weight of all metallic ores at 120,000,000 tons. The products of the quarries, clay-pits, oil-wells, phosphate-mines, and other non-metallic products, taken all together, did not in 1911 weigh in excess of 300,000,000 tons, so that the output of coal exceeds by 25 per cent. the combined tonnage of all other mineral products. The coal-mines give to

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the railroads in freight considerably more than the gross weight of all other mineral substances produced, the quantity of coal loaded at the mines for shipment in 1911 amounting to 420,000,000 short tons. The mining engineering problems in coal-mining differ somewhat from those obtaining in the metal-mines, but who is it that will say that the engineering problems in coal-mining are less serious or less difficult of solution than those encountered in metal-mining? The problems involved in the setting-up and maintaining of a proper system of ventilation alone, in the operation of a coal-mine, are sufficient to counterbalance those of square-setting, slicing, stoping, etc., in the metal-mines. The mining of coal has a direct bearing on metallurgy, for it is the coal-mines that furnish fuel for the metallurgist; and the question as to the coking quality of coals, and whether the coke is adapted to the blast-furnace, the smelter, and the foundry, puts coal-mining and metallurgy into intimate relationship. Is there in any other field of mining engineering such a close margin between cost and selling-price as in coal-mining? In 1909, the latest year for which statistics are available, the total expense of producing bituminous coal was \$395,907,026; the proceeds from the sales of the product, f. o. b. cars at the mines, was \$427,962,464, a difference of only \$32,000,000, or an average of 8.4 cents per short ton. The margin on the anthracite product in the same year was 13 cents per long ton, equivalent to 11.6 cents per short ton. In some of the gold-dredging operations the returns per cubic yard of material handled (a cubic yard being approximately equivalent to a ton in weight) may be as small, but I doubt if any other instance may be cited.

Among the mining engineers who have risen to prominence in the profession are many who have specialized in coal-mining. Among those who have passed away may be mentioned Eckley B. Coxe, R. C. Luther, W. A. Lathrop, John B. Atkinson, and among those living, such men as May, Richards, Warriner, Schluederberg, Scholz, and Stockett may be matched against their brothers specializing in other branches of the industry. For the purposes of this discussion, and in the preparation of the map which is presented herewith,¹ coal and the other fuels are included. Iron-ore, and not pig-iron, is used as the basis

¹ Impracticable to reproduce in the space available, and may be consulted at the office of the Institute.

of iron-production, while the metallic products, gold, silver, copper, lead, zinc, etc., are considered as the basis of valuation. The dollars are supposed to be laid out on squares, which are based upon the square root of the value of the mineral product in each State. Each square is divided into rectangles showing the percentages of the six or less principal products of each State, these principal products together usually representing 90 or more per cent. of the total production. The squares are superimposed on the States with their centers at approximately the centers of production in the various States. With these as a basis the center of production in the United States has been computed and is found to be in Cass county, Ill., about 120 miles nearly due north of St. Louis, and about 80 miles east of the Mississippi river from a point a short distance north of Quincy. The method of finding this center is that practiced by the Bureau of the Census in locating the center of population. An assumed meridian and an assumed parallel are taken and the distances east and west and north and south measured to the production-centers of the several States. The distances in miles are multiplied by the value of the production in each State, and the difference in totals E-W. and N-S. divided by the total production gives the corrections in miles from the assumed meridian and parallel.

At the Cleveland meeting, in October last, H. Foster Bain contended that the Secretary of the Institute should be a Western man, his idea being, I suppose, based on the erroneous supposition that from a mining standpoint the West is the more important part of the country. Drawn to its logical conclusion, this theory would result in the selection of a citizen of Cass county, Ill., whose experience in mining had been obtained from digging fresh-water clams in the bed of the Kankakee river. A study of the table appended to this paper shows that:

Coal is the principal product in 16 States, namely, Alabama, Arkansas, Illinois, Indiana, Iowa, Kansas, Kentucky, Maryland, New Mexico, North Dakota, Pennsylvania, Tennessee, Virginia, Washington, West Virginia, and Wyoming. It is the secondary product in Colorado, Ohio, and Texas. It is third in Montana and Oklahoma.

Iron-ore is the principal product in Minnesota, is second in

Alabama and Michigan, and third in Georgia, New York, Virginia, and Wisconsin.

Clay-products rank first in Connecticut, Georgia, New Jersey, New York, North Carolina, Ohio, and Oregon: are second in Indiana, Iowa, Kentucky, Maryland, New Hampshire, Pennsylvania, South Carolina, Virginia, and Washington, and third in Alabama, Illinois, Maine, and Minnesota.

Copper is the principal product in Arizona, Michigan, Montana, and Utah, and is second in Nevada, Alaska, and Tennessee.

Petroleum is the leading product of California, Oklahoma, and Texas; is second in importance in Illinois, and third in Ohio and West Virginia.

Gold holds first place in only two States, Colorado and Nevada, and in the Territory of Alaska; it is second in Arizona, California, and New Mexico, and third in Oregon.

The quarries furnish the principal mineral values in Maine, Massachusetts, New Hampshire, Rhode Island, and Vermont, all New England States; and have second place in Connecticut, Georgia, Minnesota, New York, and Wisconsin.

Lead is the leading product in Idaho and Missouri, but is relatively unimportant in other States.

Zinc is the most important mineral product in Wisconsin. It is second in importance in Missouri, and third in New Jersey.

Phosphate rock represents 95 per cent. of the mineral output of Florida, and is the most important product in South Carolina.

Sulphur is the principal product of Louisiana, closely followed by petroleum, the two comprising 90 per cent. of the State's total mineral production.

Table I. shows the percentages of the principal mineral products in the States in which mining is an industry of any importance. These percentages are based on the statistics of production in 1911. In a few cases they do not represent normal conditions; for instance, in Michigan in an ordinary year iron outranks copper, but held second place in 1911. It will be noted that although by far the largest output of coal is in the Eastern States, two of the Western States, one included in the Rocky Mountain division and one belonging to the Great Plains, have the largest percentages. Wyoming owes 91 per cent. of its production to coal and 75 per cent. of North Dakota's output is lignite. Minnesota gets recognition in having

90 per cent. of its output from its iron-mines; Arizona shows 85 per cent. of its production from copper-mines; Alaska's production is 83 per cent. and South Dakota's 92 per cent. in gold; and if it were not for 95 per cent. of phosphate rock and 95 per cent. of stone, respectively, Florida and Rhode Island would not be in the list.

TABLE I.—*Percentages of the Principal Mineral Products of the United States.*

ALABAMA.	Per Cent.	IDAHO.	Per Cent.
Coal.....	68.	Lead.....	60.
Iron.....	17.	Silver.....	25.
Clay.....	7.	Gold.....	8.
Stone.....	3.3		
Cement.....	1.9	ILLINOIS.	
ALASKA.		Coal.....	56.
Gold.....	83.	Petroleum.....	18.5
Copper.....	14.	Clay.....	14.
ARIZONA.		Stone.....	3.4
Copper.....	85.	INDIANA.	
Gold.....	8.	Coal.....	41.
Silver.....	4.	Clay.....	19.
ARKANSAS.		Cement.....	16.
Coal.....	58.	Stone.....	14.
Stone.....	1.	IOWA.	
CALIFORNIA.		Coal.....	60.
Petroleum.....	43.	Clay.....	20.
Gold.....	22.	Cement.....	9.
Cement.....	9.6	Gypsum.....	4.
Clay.....	5.4	KANSAS.	
Copper.....	5.	Coal.....	39.
Stone.....	5.	Natural gas.....	20.
COLORADO.		Cement.....	14.
Gold.....	37.	Clay.....	9.
Coal.....	28.4	KENTUCKY.	
Zinc.....	9.4	Coal.....	72.
Stone.....	7.6	Clay.....	12.5
Lead.....	5.3	Stone.....	6.5
CONNECTICUT.		LOUISIANA.	
Clay.....	40.	Sulphur.....	45.
Stone.....	40.	Petroleum.....	45.
FLORIDA.		MAINE.	
Phosphate rock.....	95.	Stone.....	49.
GEORGIA.		Lime.....	17.
Clay.....	43.	Clay.....	13.
Stone.....	32.	Mineral waters.....	9.
Iron.....	6.	MARYLAND.	
Coal.....	4.	Coal.....	55.
		Clay.....	20.
		Stone.....	12.

142 GEOGRAPHICAL DISTRIBUTION OF MINING DEVELOPMENT.

MASSACHUSETTS.		NORTH CAROLINA.	
	Per Cent		Per Cent
Stone.....	55.	Clay.....	46.
Clay.....	26.	Stone	10.
Lime.....	10.	Mica.....	8.
MICHIGAN.		NORTH DAKOTA	
Copper.....	42.	Coal.....	75.
Iron.....	37.		
Cement.....	5.	OHIO.	
Coal.....	4.	Clay..	34.
Clay.....	3.	Coal	33.
MINNESOTA.		Petroleum.....	9.8
Iron.....	90.	Natural gas.....	9.6
Stone.....	3.	Stone.....	6.
Clay.....	3.		
MISSOURI.		OKLAHOMA.	
Lead.....	31.	Petroleum.....	62.
Zinc.....	26.	Natural gas.....	16.
Coal.....	12.	Coal.....	14.8
Clay.....	12.		
Cement.....	6.	OREGON.	
Stone.....	4.	Clay..	38.
MONTANA.		Sand and gravel.....	20.
Copper.....	64.	Gold.....	20.
Silver.....	13.	Stone.....	18.
Coal.....	10.		
Gold.....	7.	PENNSYLVANIA.	
NEVADA.		Coal.....	77.
Gold.....	53.	Clay.....	5.
Copper.....	24.	Cement	5.
Silver.....	21.	Natural gas.....	4.5
NEW HAMPSHIRE.		Petroleum.....	2.5
Stone.....	60.		
Clay.....	25.	SOUTH CAROLINA.	
NEW JERSEY.		Phosphate rock.....	37.
Clay.....	66.	Clay.....	37.
Cement.....	12.	Stone.....	18.
Zinc.....	7.		
Stone.....	6.	RHODE ISLAND.	
NEW MEXICO.		Stone.....	95.
Coal.....	62.		
Gold.....	10.	SOUTH DAKOTA.	
Silver.....	9.	Gold.....	92.
NEW YORK.			
Clay.....	25.	TENNESSEE.	
Stone.....	16.	Coal.....	40.
Iron-ore.....	7.	Copper.....	13.
Cement.....	7.	Phosphate rock.....	10.
		Stone.....	8.5
		Clay.....	7.5
		TEXAS.	
		Petroleum.....	35.
		Coal.....	17.
		Clay.....	14.
		Natural gas.....	6.

UTAH.		WASHINGTON.	
	Per Cent.		Per Cent.
Copper.....	43.	Coal.....	58.
Silver.....	15.	Clay.....	18.
Lead.....	12.	Stone.....	10.
Gold.....	11.	Cement.....	9.
Coal.....	10.	WEST VIRGINIA.	
VERMONT.		Coal.....	53.
Stone.....	75.	Natural gas.....	28.
Slate.....	20.	Petroleum..	10.
VIRGINIA.		Clay.....	4
Coal..	46.	WISCONSIN.	
Clay.....	13.	Zinc.....	30.
Iron-ore.....	8.	Stone.....	20.
Stone.....	8.	Iron-ore.....	11.
Soapstone.....	5.	Clay.....	9.
		Lead.....	9.
		WYOMING.	
		Coal.....	91.

TABLE II.—Principal Products of the Leading Mining States in 1911.

State.	Coal.	Copper.	Clay-Products.	Petroleum.	Gold.	Iron-Ore.	Stone.	Natural Gas.	Cement.
Pennsylvania.....	\$521,587,250	\$82,703	\$20,270,083	\$10,894,071	\$539,553	\$8,147,505	\$18,010,796	\$19,506,319
Illinois.....	99,310,478	19,731,339	19,731,339	3,167,930	687,726	3,588,301
West Virginia.....	59,670,515	4,383,420	12,767,293	1,106,012	28,151,907
Ohio.....	81,810,123	32,663,895	9,479,512	30,519	5,796,849	9,367,347	1,218,109
California.....	16,097	4,179,456	1,915,866	\$19,928,500	1,676,942	800,714	8,737,150
Michigan.....	2,791,461	27,273,155	2,083,932	a 40,823,501	23,808,935	1,065,530	1,330	3,024,676
Minnesota.....	1,093,478	48,147,760	1,702,925
Montana.....	34,974,811	30,547	3,649,700	212,233	3,349,312
Missouri.....	5,342,108	40,051	6,274,533	153,676	2,338,585
Colorado.....	6,481,066	1,238,983	1,096,709	228,101	19,188,800	1,514,827
Arizona.....	14,747,764	37,900,317	106,882	3,521,400	65,711
Oklahoma.....	756,689	26,451,767	795,879	6,731,770
New York.....	6,291,491	17,702,527	10,161,576	1,246,950	4,186,200	2,039,009	6,895,466	1,418,767	2,812,440
Utah.....	4,248,636	538,955	215,037	827,523
Indiana.....	15,826,895	8,195,127	7,000,771	1,223,835	4,413,655	1,192,118	5,937,241
Nevada.....	18,006,900
Alabama.....	19,079,919	1,947,102	18,400	4,876,106	923,998	529,359
New Jersey.....	18,178,228	1,158,271	1,997,410	3,250,528
Kansas.....	9,645,572	2,360,262	608,756	803,222	4,951,531	3,725,108
Iowa.....	12,063,507	4,432,874	736,507	70	1,881,255
Alaska.....	7,200	2,789,361	16,665,200	a	a
Kentucky.....	18,617,217	2,268,004	328,614	78,212	1,227,009	107,689
Texas.....	3,273,288	13	2,659,919	6,554,552	3,900	588,777	1,014,945	a
Tennessee.....	7,209,731	2,370,643	1,385,100	11,900	632,359	1,198,798	300	a
Idaho.....	4,808	564,285	a	1,337,900	61,250
Washington.....	8,174,170	24,458	2,840,372	840,000	1,679,872	1,406,807
Virginia.....	6,294,804	1,739,900	5,668,814	3,100	1,146,188	821,798	a	a
Louisiana.....	531,949	a
Wisconsin.....	1,158,139	21,300	1,386,616	2,875,102
Wyoming.....	10,508,863	16,312	77,146	a	479,922	40,514	a
Florida.....	5,197,066	2,944	217,535	a
Maryland.....	86,466	1,133,339	a
Vermont.....	201	61,365	6,145,351
South Dakota.....	357,550	174,651	7,429,500	148,190
New Mexico.....	4,525,925	1,700,287	761,700	146,401	406,451	16,984
Massachusetts.....	2,636,380	3,691,747
Georgia.....	246,208	1,967,077	380,186
Arkansas.....	8,396,849	480,643	32,000	315,701	526,352	a
Other States.....	828,826	24,235	8,594,468	132,232	923,600	260,589	8,327,231	1,170,297	6,056,791
Total.....	\$626,306,876	\$137,154,092	\$162,236,181	a \$136,149,173	\$96,890,000	\$86,419,880	\$77,108,567	\$74,127,584	\$66,705,136

a See Notes under "Remarks."

TABLE II.—Principal Products of the Leading Mining States in 1911—(continued).

State.	Lead.	Silver.	Zinc.	Sand and Gravel.	Lime.	Miscellaneous.	All Other.	Total.	Remarks.
Pennsylvania	\$3,025,267	\$2,685,574	\$9,610,469	\$414,112,573	
Illinois	\$27,720	1,990,922	123,762	Flint spar .. \$181,635	1,674,165	106,581,165	
West Virginia	\$28,776	552,388	636,600	Salt .. 78,805	171,298	101,918,248	
Ohio	2,104,776	1,007,424	Quicksilver .. 1,109,463	1,831,137	97,000,284	
California	55,350	686,300	109,326	116,384	564,175	Salt .. 2,573,126	3,139,500	90,517,506	<i>a</i> Includes Asphalt
Michigan	274,100	565,969	352,045	Gypsum .. 673,871	828,365	65,077,292	
Minnesota	399,989	212,968	Gypsum .. 221,613	973,871	53,469,561	
Montana	224,910	6,968,500	2,621,110	2,461	18,480	Gems .. 81,380	53,451,926	93,451,926	
Missouri	16,308,630	26,500	14,339,560	1,012,674	722,563	Barytes .. 81,380	1,197,998	52,636,348	
Colorado	2,755,890	3,968,800	4,314,562	65,366	31,614	Tungsten, Uranium, and Vanadium	1,332,307	51,958,299	
Arizona	310,500	1,743,600	284,264	69,910	Gems .. 17,613	96,471	41,104,731	
Oklahoma	174,250	566,156	97,539	11,003	Gypsum .. 293,302	816,045	42,678,446	
New York	4,967,620	6,280,500	798,156	2,414,452	524,843	Gypsum .. 1,199,596	12,731,273	12,392,189	<i>b</i> Chiefly Aluminum
Indiana	23,859	74,770	Asphalt .. 516,614	335,498	41,139,075	
Nevada	33,501	824,950	Min. waters .. 6,53,641	218,039	37,430,187	
Alabama	97,470	7,130,100	181,880	215,413	300,787	122,783	31,116,101	
New Jersey	1,724,502	1,058,926	113,784	Min. waters .. 210,129	1,009,746	27,010,278	
Kansas	226,980	780,102	104,058	3,440	Salt .. 806,027	2,58,384	24,087,806	
Iowa	393,649	80,914	Gypsum .. 47,270	390,850	21,107,106	
Alaska	4,500	252,900	15,121	Flint spar .. 96,574	358,663	18,910,731	<i>a</i> Included in "all other."
Kentucky	24,370	389,368	Gypsum .. 52,409	20,072,420	<i>a</i> Included in "all other."
Texas	5,130	239,900	543,866	218,007	Flint spar .. 491,685	2,136,137	18,817,304	<i>a</i> Included in "all other."
Tennessee	57,800	114,000	518,512	293,763	Asphalt .. 786,785	2,132,245	17,902,451	<i>a</i> Included in "all other."
Idaho	10,514,310	4,419,800	362,520	38,088	Phos. rock .. 1,688,207	232,399	17,583,340	<i>a</i> Included in "all other."
Washington	54,090	124,400	1,368	838,400	2,8,363	50,706	15,833,556	<i>a</i> Included in "all other."
Virginia	36,000	100	25,992	204,170	483,016	Talc .. 690,926	1,674,876	13,600,304	<i>a</i> Included in "all other."
Louisiana	844,281	Pyrites .. 535,491	12,710,958	<i>a</i> Included in "all other."
Wisconsin	357,600	400	21,662	961,553	Min. waters .. 935,988	6,163,914	12,151,436	<i>a</i> Included in "all other."
Wyoming	38,546	Gypsum .. 16,238	167,709	11,486,982	<i>a</i> Included in "all other."
Florida	33,339	49,221	Phos. rock .. 2,655,571	10,950,928	<i>a</i> Included in "all other."
Maryland	246,486	362,839	Fulcrs' earth .. 1,624,911	671,555	9,586,663	<i>a</i> Included in "all other."
Vermont	62,540	191,081	State .. 1,624,911	334,183	8,431,516	<i>a</i> Included in "all other."
South Dakota	2,970	108,200	41,296	37,673	191,103	8,037,372	
New Mexico	124,230	724,300	430,692	18,871	13,004	186,177	7,869,918	
Massachusetts	222,822	605,662	312,550	6,623,077	
Georgia	300	175,734	204,067	413,711	6,171,367	
Arkansas	1,350	45,890	183,897	109,067	5,829,006	<i>a</i> Included in "all other."
Other States	189,130	26,640	1,570,509	1,570,362	25,365,391	<i>b</i> Largely Bauxite.
Total	\$36,553,340	\$32,615,700	\$30,964,794	\$21,158,573	\$13,680,051	\$27,916,320	\$51,836,269	\$1,677,411,459	<i>a</i> Includes Asphalt in California.

b See Remarks.

DISCUSSION.

A MEMBER:—I would like to call attention to the fact that the speaker has failed to indicate where is the center of gravity of the membership of the Institute, while he is dealing with this map.

MR. PARKER:—I think you will find that a little further east.

CHAIRMAN JAMES F. KEMP:—In connection with Mr. Parker's paper, I would like to inject a little matter of history into our program here, as regards the founding of the Institute. The credit for the founding of the Institute really applies to miners who were engaged in the anthracite coal region. The three men who signed the circular for the call for the meeting which brought the engineers to Wilkes-Barre in May, 1871, were Eckley B. Coxe, Richard P. Rothwell, and Martin Coryell.

MR. PARKER:—I have mentioned in the paper, where it comes to the question of mining engineering, that I think that those who have specialized in coal mining have made just as much of a reputation as those who have been more prominently identified with metal mining. We can point to Eckley B. Coxe and Martin Coryell, and our friend Lathrop, and many others, and put them side by side with men who have made their reputations in the mining of gold and silver, and I do not think they will suffer by the comparison.

A Problem in Mining, together with Some Data on Tunnel-Driving.

BY F. M. SIMONDS AND E. Z. BURNS, NEW YORK, N. Y.

(New York Meeting, February, 1913.)

The Problem in Mining.

The Rawley property is located in the Kerber Creek mining-district, Saguache county, Colorado, at an elevation of 10,600 ft. (See sketch-map, Fig. 1.) Ore was found in this vicinity as early as 1880, and claims covering a large area were located and in many instances patented. The ores, however, being of comparatively low grade, and the nearest railway station, Villa Grove, being some 20 miles away, it is extremely doubtful whether any considerable amount of ore was ever handled at a profit. It is safe to say that several hundred thousand dollars were spent (a large part in building mills) previous to 1902-03.

About that time a very capable promoter gathered together a group of these claims and interested some New York people, who had had no previous mining experience, in the purchase of these claims. Following the organization of a company, this promoter became General Manager. As often happens under such conditions, the first step was to erect a mill to treat 100 tons per day. It may not be fair to this enterprise to say that the failure which followed might have been foretold, simply on general principles, but this much is certain—the water supply fluctuates in the course of a season through a wide range and for a considerable portion of the year was inadequate to run this mill at capacity. Further, it is currently reported that some of the mill foundations were placed on ice. However true this statement may be, it is a fact that, when first inspected by us, many of the belts in the mill were held on their pulleys by means of idlers. A shut-down followed, with a material re-adjustment of interests, at which time we became consulting engineers for the company.

The question which then confronted the owners was, briefly, whether to abandon the enterprise as a total loss, or to pursue a course of systematic development of the ore body as exposed by three short drifts.

Without committing themselves to an extensive policy of development, the stockholders finally decided to do a small amount of development-work—to put some money into the ground. Almost from the start, this work showed encouraging results in the way of blocking out ore of a satisfactory grade, although it was conducted during the next few years under many discouragements. Besides the natural difficulties of doing

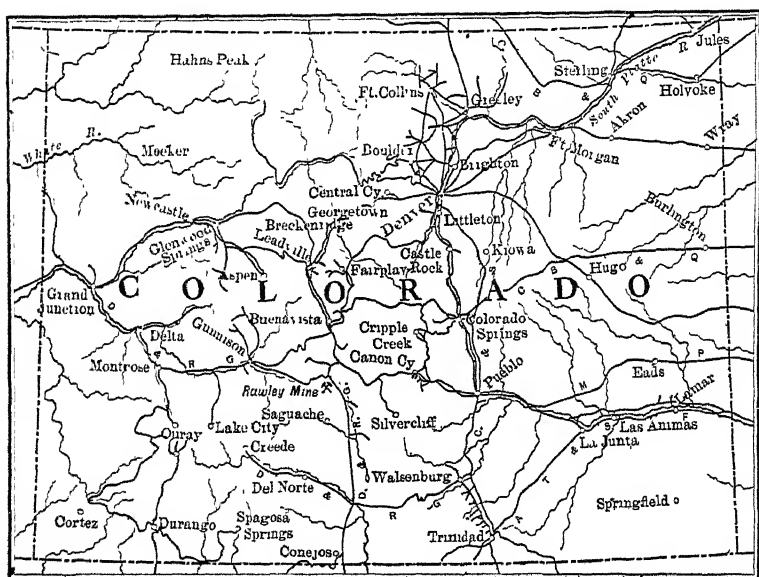


FIG. 1.—SKETCH-MAP OF COLORADO, SHOWING THE LOCATION OF THE RAWLEY MINE.

work at such a distance from the railway, there were frequent labor disturbances, causing a shut-down, in one instance, of nearly two years. This brings us to 1910, at which time we had proved the vein to a depth of 600 ft., and had developed in "blocked out" and "probable" ore a gross value of about \$5,000,000. We found that the water was increasing at the rate of about 100 gal. per min. for each 100 ft. of added depth. Also, we found the character of the ore had changed, as we had anticipated in a report several years before, from a silver-lead-copper

concentrating-ore in the upper levels, to a semi-pyritic silver-copper ore on the sixth level.

In view of the altered problem of mining and treatment which this new condition presented, we concluded that it would be very desirable to block out a large tonnage of this silver-copper ore, in order, first, to prove the average character of ore to be encountered with depth, and, second, to obtain ore to which the silver-copper concentrates derived from the silver-lead-copper ore of the upper levels could be added and more economically treated.

The following summary of the facts so far demonstrated by our work and study will aid to a more complete understanding of the problem which now confronted us:

1. The country-rock for several miles in each direction is of igneous origin, and all of strictly surface types—either tuffs, felsites, pumices, amygdaloids, simple andesites, or flowage-breccias. To the north, east and west, in various gulches, rock of this character is exposed at elevations of from 1,000 to 2,000 ft. lower than our 6th-level workings.

2. Across the country from east to west, cutting the Rawley vein nearly at right angles, is a zone of faulting some 50 ft. wide, dipping at an angle of about 55° S., as shown in Fig. 2.

This faulting was subsequent to the vein-formation, and the zone incloses, at various places, bunches of ore detached from the veins which it intersected. The lateral throw of this fault was slight, if any, but there is reason to believe that the vertical throw must have been considerable. However, no definite data existed at the time, and only meager data have since been developed to indicate the amount of this vertical displacement. There being but little lateral throw, the alignment of vertical or nearly vertical veins was only slightly disturbed by this faulted zone.

3. The Rawley vein is nearly vertical, extends in a general N-S. direction, and would undoubtedly be classed as a fissure. The walls are well defined, and no evidence of any fault, except the one above described, appears on the surface for a long distance on either side. The vein was developed more or less on six levels, as shown in Fig. 2.

4. We made many sections of the country-rock from various levels and from the surface for microscopical study. This work

was done by Dr. Charles Berkey, of the Geological Department, Columbia University. Generally speaking, the rock consisted of a series of igneous flows. A very important point which we wished to determine was the general dip and strike of these flows. The difficulty of finding a true bedding-plane and distinguishing between it and a fracture was great. We finally obtained a sample of rock properly oriented on the ground, which gave us approximately this information. We also had many sections of the ore from various levels made for microscopic study. The results of this were very instructive, and

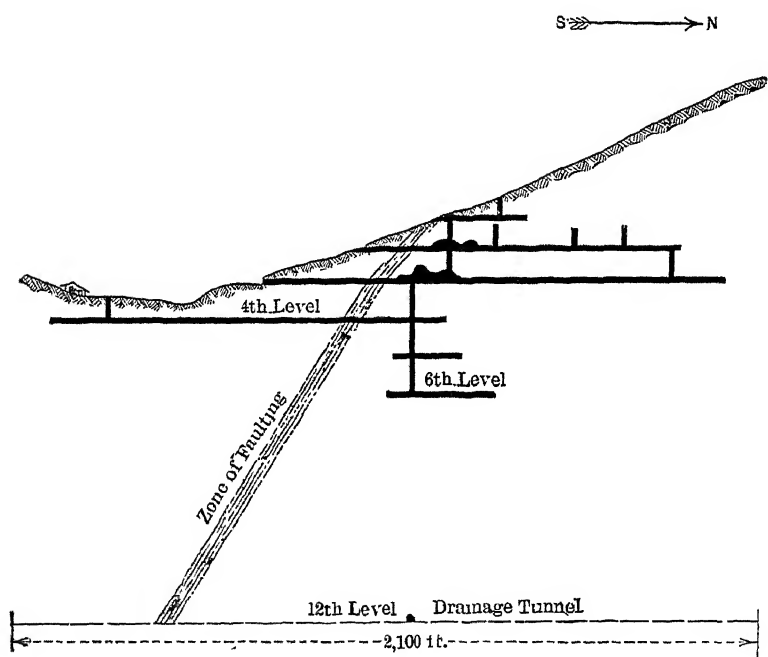


FIG. 2.—SECTION ON THE VEIN OF THE RAWLEY MINE.

showed, first, that there was no indication of secondary enrichment, and, second, that the governing factor in the deposition of the minerals, as the evidence seemed to indicate, was the hydrostatic pressure of the mineralized water at different depths. Thus, galena predominated in the upper levels and gradually gave place to iron and copper as depth was obtained. Assuming this to be the correct theory, we reasoned that there was a probability of the increase of gold- and silver-values with greater depth. (We may here add that this has actually proved

true. On the 12th level, the average gold- and silver-content has more than doubled.)

5. Surveys showed that the property was only about 6.5 miles in an air-line from a point on the railway at which an adequate water supply was found. According to careful estimates, Rawley ore could be handled at a good profit on a scale of from 200 to 300 tons per day to this point, and the ore blocked out above the 6th level would net considerably over \$1,000,000.

6. Surveys showed also that a tunnel some 6,200 ft. long would intersect the Rawley vein (if it was there) on the 12th level, 600 ft. below our lowest workings.

With the above facts as a basis, two courses were open: to continue sinking, or to drive a tunnel.

Several years' experience in sinking, or trying to sink, had demonstrated that the cost of fuel and supplies was excessive, that the supply of labor for sinking a wet shaft was extremely fluctuating, as a rule inadequate, and always inefficient (good men do not have to locate in a remote camp); that pumping had to be continuous, whether progress was being made or not; in short, that the most liberal estimate for cost had to be multiplied by x to the n th power.

The alternative plan of driving a tunnel offered certain marked advantages. The cost of tunnel-driving could be estimated with a fair degree of accuracy; we were firmly convinced that the vein would be found in place, with perhaps a slight flattening towards the east; we were very sure that, if the vein were encountered, the upper workings would drain without the necessity of an up-raise; whether we encountered the vein or not, the decreased cost of developing and extracting the ore above, due mainly to the elimination of continuous pumping, would fully pay for the tunnel; and, finally, the tunnel would serve to develop certain veins, showing on the surface, in claims owned by the company along the proposed course.

The oft-repeated maxim, "stick to your ore," is to the mining engineer almost what the theory of the conservation of energy is to the mechanical engineer; and the members of the Institute can understand with what reluctance we finally advised our clients contrary to such a generally accepted and safe mode of

procedure. However, we believed that at a stage in the development of a mining property such as we had reached, we were warranted, after having given long, close, and intelligent study to the characteristics of the ore deposit, in offering advice on economic grounds, without giving such overwhelming weight to the question of the continuity of the ore body. Realizing our great responsibility, however, we suggested the engagement of some prominent and unbiased engineer to go over all the facts and conditions which we could submit, and pass on our conclusions.

This suggestion was adopted, and, after careful deliberation, the engineer reported. His conclusions, which, unlike those of physicians in consultation, were adverse to ours on practically every important point, may be fairly summarized as follows: The rock-formation (igneous flows) would change at or before reaching the depth of 1,200 ft., and the vein would probably not be found at that depth, or, if found, would probably be barren, or of too low-grade to work at a profit; the tunnel would not drain the upper workings without a very expensive up-raise; and altogether the risk was far too great to warrant so large an expenditure.

It is apprehended that these conclusions would have been approved by a large number of members of the Institutè. It is not in any way a reflection on his recognized great ability, but only an illustration of the great difficulty that one engineer must find in giving due weight to the study and experience of another engineer, that he proved to be in error on every important point. The company finally assumed the risk of running the tunnel; and the results verified the correctness of our deductions.

THE TUNNEL.

Except the preliminary surveying and the placing of compressor-foundations, preparations for driving this tunnel began April 1, and actual work at the portal, May 7, 1911.

The first work was to establish, near the portal, camp-buildings suitable for the accommodation of from 30 to 50 men. In order to secure good men and keep them in a place like this, two things are very essential: clean, comfortable quarters; and an abundance of the best food obtainable. No reasonable expense was spared in order to meet these requirements.

In a general way we planned to locate the power house, blacksmith shop, tool house, etc., as close to the portal as practicable, and so arranged as to involve the minimum amount of labor in handling steel, pipe, tools, etc. The bunk houses and boarding house were placed as nearly on a level with the portal as possible, and just so far away that the men when off shift could sleep undisturbed by the handling of steel, dumping of rock, and other noises. We may seem to be giving undue prominence to these little matters; but, in our opinion, attention to these and similar small details saved thousands of dollars during the course of the work.

The power plant consisted of one 80-h-p. return tubular boiler, with one 40-h-p. return tubular boiler in reserve; one Rand Imperial type No. 10 compressor (all of which were on hand); one No. 3 Roots blower (capacity 7.75 cu. ft. per rev.) direct connected to a 9- by 9-in. upright engine.

For drilling, we used two No. 8 Leyner drills, with cross-bar, keeping one extra drill (and later on, two) in reserve, and maintaining also an abundance of extra parts always on hand. In the blacksmith shop we had one No. 2 Leyner steel-sharpening machine, operated by compressed air at 100 lb. pressure. Next to the steel-sharpener was placed a No. 8 Bradley coke-furnace for heating the steel, operated by a steam-driven turbine-blower, 12 in. in diameter, and furnishing at normal speed about 8 oz. pressure.

The cross-section of the tunnel was made 7 by 8 ft. in the clear, and where timbering was necessary the same dimensions were maintained inside the timbers. The arrangement of track, ditch, and pipes is shown in Fig. 3.

The grade of the tunnel was fixed at 0.5 per cent. For track, 18-in. gauge, 16-lb. steel rails were used, placed on ties 2.5 ft. apart from center to center. At frequent intervals, ties were placed across the ditch to support the pipes. Switches were provided at intervals of about 1,500 ft. for the passage and storage of cars. For fuel, wood was purchased from the Forest Reserve, at points so located that the haul to the tunnel portal would be down-grade. For blasting, 40 per cent. dynamite was used almost exclusively.

The tram cars used were 4 by 2 by 2 ft., body of 0.25-in. steel, with double bottoms, boxes reinforced around rims and

centers with 0.5- by 2-in. steel, and with turn-table dump, mounted on chilled iron wheels. The cars had a continuous draw-bar and were coupled by means of three long links, giving considerable slack for facilitating the starting of a train. Mules were used for tramming, and by reason of this slack between the cars, one mule could start a train of 8 or 10 cars.

The labor for this work was organized as follows:

	Rate of wages per day
One superintendent	
One surveyor-bookkeeper	
One foreman.....	\$6.00
One shift-boss per shift.....	5.00
Two machine men per shift	4.50
Two machine helpers per shift.....	3.75
Three to four muckers per shift	3.50
One swamper per shift.....	3.00
One to more pipe-, track- and ditch-men, as required.....	3.50
One trammer per shift.....	3.50
One blacksmith.....	4.50
One head mechanic.....	4.50
Two power-house men (two shifts).....	4.00
One car-greaser, dump-, and powder-man per shift.....	3.00
One wood-hauler.....	3.00
Timber-framer, as required...	4.00
One roustabout.....	3.00
One cook and helper	

The above gives a fair representation of the working-force, and, with uniform rock-formation, little change in number or arrangement would be required. Unfortunately, however, the rock proved to be anything but uniform. At any moment, the andesite might change from a very soft material, full of slips, and requiring to be timbered close to the face, to a very tough, hard rock, highly silicified. This silicified rock, largely jasper, carried sometimes 30 per cent. of pyrite, and would eat up steel faster than the blacksmith could get it sharpened. For instance: one day so much muck would be brought down that we could not put muckers enough in the tunnel to get it out on time, while, perhaps, on the next day, two or three muckers per shift would be ample.

This irregularity of the rock-formation had far-reaching effects. By reason of it, no system of drill-holes was possible, and, from day to day, each shift had to lay out the holes in such a way as seemed best to the foreman or the shift-boss.

Moreover, in the use of powder, no attempt could be made to determine a point of maximum efficiency, as would have been possible with uniform ground.

The sequence in the operations of driving the tunnel, when everything was in proper running order, was about as follows:

Just before shooting, the compressed air was turned on through a small nozzle pointed directly at the place where the muck pile would be. At the same time the blower was speeded

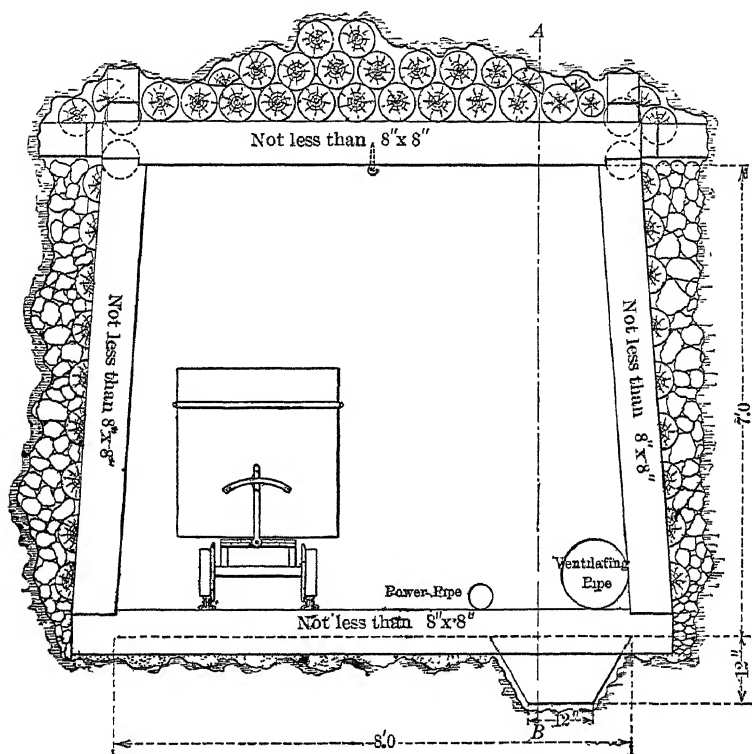


FIG. 3.—CROSS-SECTION OF TUNNEL, SHOWING ARRANGEMENT OF TRACK, DITCH, AND PIPES.

up. After a round had been fired, the blower cleared the air by suction from the face sufficiently to allow the men to begin work, an operation requiring about 20 min. The foreman and drill-men would then begin picking down roof and face. When the roof was in a safe condition, the drillers, or helpers, would "muck back" sufficiently to allow the bar to be set up in its upper position. This mucking, as a rule, took a few minutes only, and simply amounted to trimming off the top of

the muck pile. During this time and while the bar was being set up, helpers would be connecting up hose and muckers bringing up empty cars. The hose was never allowed to lie along the bottom of the tunnel and on the muck pile, where it would be constantly exposed to damage, and would interfere more or less with the mucking. To avoid this we put near the roof, at frequent intervals, cross-pieces over which the hose was hung. The cross-pieces consisted of two pieces of pipe, telescoped, and with a set-screw, permitting them to be put up and taken down with great facility. In order to have the bar set up each time at the same height above the grade-line of the tunnel, iron hangers were hung from the last two survey points, for example, 100 and 300 ft. from the face, respectively. By means of wire at the top, these hangers were adjustable; and, as a hanger was moved forward to a new station, its length was calculated exactly, so that its bottom, shaped like a stirrup and carrying a candle, would be at the same height above the grade-line as the bar was required to be set. As the survey points were exactly along the center-line of the roof, the foreman, by standing back and sighting towards the face, could, in about a minute, signal to the driller at the face both the center-line and the height at which to set the bar. This operation was performed before each set-up, and no difficulty was experienced in keeping the alignment and grade almost perfect.

As soon as the condition of smoke permitted, mucking began. The sequence of the various operations of mucking was about as follows:

Empty train arrived and was pulled as near to muck pile as practicable.

Empties were turned on sides.

Down-tunnel end-car of empties righted on track, and pushed to muck pile.

Mucking began.

Full car pushed down beyond last empty.

Next empty car righted and pushed to muck pile.

Same order as above was followed, until train of empties was all loaded

Another train of empties arrived and was hauled to a short distance from loaded train.

Mule was detached from train of empties and hauled loaded train back towards muck pile as far as practicable.

Train of empties was hauled as near loaded train as practicable.

Empty train was turned over on side.

Loaded train started for portal.

THE RAWLEY MINING COMPANY.

DAILY REPORT.

No. 12 Tunnel.

	Shift Bosses.	Machine Men.	Machine Helpers.	Number of Holes.	Total Feet Drilled	Actual Drilling Time.	Total Working Time, Machine Men.	Total Working Time, Muckers.	Rounds Shot.	Missed Holes.	Distance Gained.	Sticks Powder Used.	Approximate Weight of Powder in Pounds	Number of Caps	Feet of Fuse	Muckers	Swampers.	Trammers	Animals Employed Underground.	Cars Trammed.	Pipe, Track and Ditch Men.	Timbermen	Timber Helpers	Distance Timbered.	Car (Pressers, Dump and Powder Men.	Blacksmiths.	Boiler and Compressor Men	Woodmen and Timber Runners	Teamsters.	Other Laborers (Miscellaneous)	Wood Consumed For 24 Hours.										
7 A. M. Shift																																									
3 P. M. Shift																																									
11 P. M. Shift																																									
Totals.....																																									
Number of Employees Underground.																																Remarks :—									
“ on Surface																																									
Total Crew (exclusive of boarding house and office)																																									
Distance Gained for Month																																Feet.									
Total Length of Tunnel																																“									
Distance Timbered for Month																																“									
Total Timbering in Tunnel																																“									
Total Wood Consumed for Month																																Cords									
Total Powder “ “ 4” per cent																																Pounds									
Total Caps “ “ “																																“									
Total Fuse “ “ “																																Feet.									
																																Manager.									

FIG. 4.—FORM OF DAILY TUNNELING REPORT.

The foregoing data give the full cycle of events as regards mucking; but perhaps a few additional remarks may be appropriate. When the rock breaks well, and the advance is regular and rapid, the rate of advance becomes a question of disposing of the muck. Hence the system as above outlined must be carried out with almost mechanical precision. It is here that attention

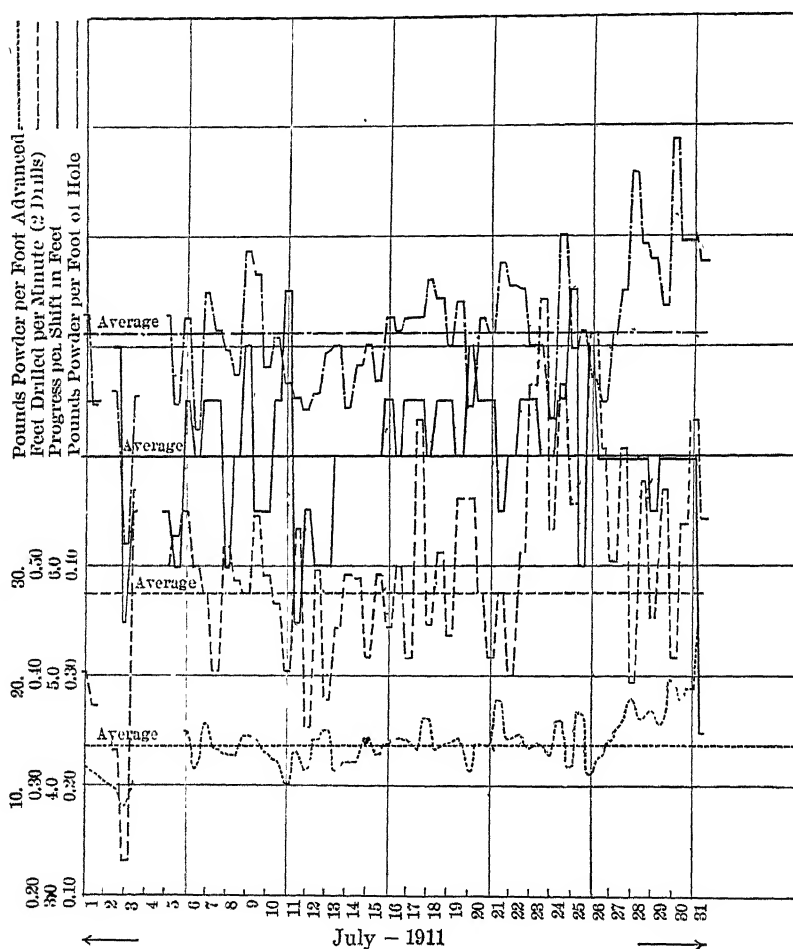


FIG. 5.—DIAGRAMMATIC CHART OF OPERATIONS FOR JULY, 1911.

to small details is of great importance. Consider, for instance, the question of whether men are right-handed or left-handed. Suppose you have four men shoveling into the car. The ideal condition in that case is to have two right-handed and two left-handed men. Located so far from an adequate labor supply,

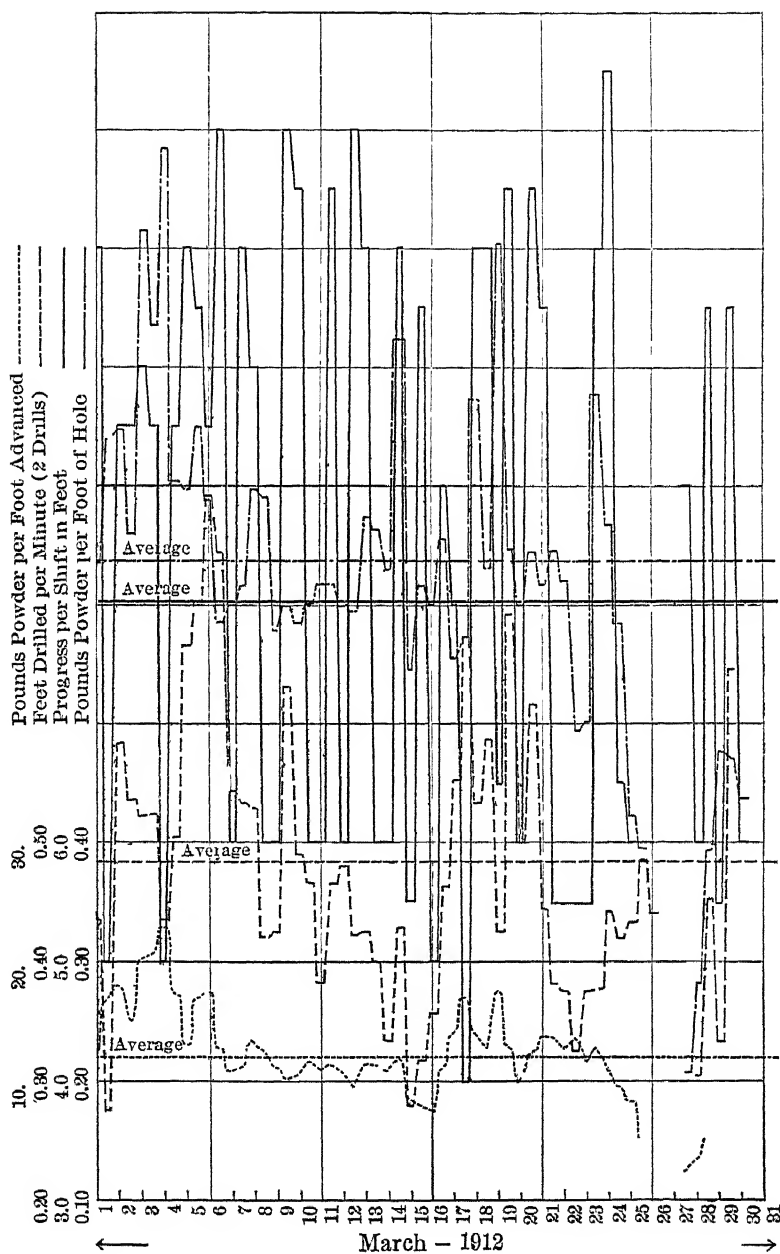


FIG. 6.—DIAGRAMMATIC CHART OF OPERATIONS FOR MARCH, 1912.

we found it, at times, impossible to fulfill this condition; but it was constantly kept in mind when employing new men.

Beginning June 14, 1911, and continuing throughout the course of the work, a daily report, shown in Fig. 4, was sent to the New York office. From these daily reports various details were calculated and the data tabulated for each month. As 17 months of these tabulated data seems an unnecessary mass of figures to present, we have selected the months of July, 1911, and January, March and July, 1912, as representing typical periods under varying conditions. (See Table I.) In addition, we present the totals and averages for all the months. (See Table II.) From the tabulated data for July, 1911, and March, 1912, was made the efficiency-chart, Figs. 5 and 6. It is with regret that we call attention to the omission of one item that could easily have been supplied, viz.: The number of drills dulled per round. However, after the work was under way and some little time had gone by, we did not like to introduce a new item into the already lengthy daily report.

As hereafter explained, we used, after March, 1912, three full shifts in driving the tunnel. In tabulating the data for a continuous chart, we combined the results of two shifts in order to maintain the uniformity of our chart. A simple inspection will show, as a rule, where this combination is made. Of course, the chart does not really show anything more than the figures show, but, from an examination of the chart, one can see better at a glance the relation of the various factors given.

It was our idea to have one line on our chart represent, if we may use the expression, the "tunneling quality" of the rock. For this purpose we had a record kept of the data which would permit us to calculate, for each round of holes, the distance in feet drilled per minute. Of course this is only one of many factors that would be necessary in order truly to represent "tunneling quality." We believe equally good data to represent this line could have been obtained by keeping a record of the number of feet drilled per round, and the number of drills dulled per round, with the advantage of possibly greater accuracy in the record. However, it is interesting to note that the months when our average in "Feet drilled per Minute" was high, were quite uniformly good months, both in progress and general efficiency.

TABLE I.—Data Compiled from Daily Reports.

July, 1911. (2 drills).	No. of Holes.	Total Feet Drilled.	Actual Time Drilling, Hours	Lb. Powder per Round	Advance Gained Feet.	Feet Drilled per Minute.	Lb. Powder per Ft. of Drill-Hole.	Feet of Drill-Hole per Foot Ad- vanced.	Lb. Powder per Foot Advanced.	Average Time of Drilling One Ft. in Minutes.	No Rounds
1	24	170	7.0	90.	7.5	0.405	0.529	22.5	12.0	..	1
	23	190	8.5	85.	7.5	0.378	0.447	25.3	11.3	..	1
2	18	108	10.	0.180	1
	22	180	9.	82.5	8.	0.333	0.458	22.5	10.3	..	1
	21	140	10.	45.	5.5	0.233	0.321	23.6	8.2	..	1
3	22	154	4.5	70.	6.5	0.570	0.455	23.7	10.8	..	1
	23	180	6.	95	6.5	0.500	0.528	27.7	13.9	..	1
5	23	190	6.	85.	6.0	0.528	0.447	31.2	14.2	..	1
	27	214	6.5	112.5	7.5	0.549	0.526	28.5	15.0	..	1
6	25	195	6.5	82.5	7.0	0.500	0.423	27.9	11.8	..	1
	27	214	7.5	117.5	7.5	0.476	0.549	28.5	15.7	..	1
7	25	195	8.	100.	7.5	0.406	0.513	26.0	13.3	..	1
	23	156	5.	77.5	6.0	0.520	0.497	26.0	12.9	..	1
8	23	190	6.5	90	7.0	0.487	0.474	27.1	12.9	..	1
	23	200	7	117.5	8.0	0.476	0.587	25.0	14.7	..	1
9	22	164	5.	92.5	6.5	0.547	0.564	25.2	14.2	..	1
	25	177	6.	85.	6.5	0.492	0.480	27.2	13.1	..	1
10	22	182	6.5	92.5	7.5	0.467	0.508	24.3	12.3	..	1
	22	182	7.5	85.	8.5	0.405	0.467	21.4	10.0	..	1
11	20	160	5.	72.5	5.5	0.533	0.453	29.1	13.1	..	1
	23	170	8.	75.	6.5	0.354	0.441	26.2	11.5	..	1
12	21	161	5.5	85	6.0	0.497	0.457	27.3	14.2	..	1
	25	182	8.	90.	6.0	0.379	0.494	30.3	15.0	..	1
13	21	160	6.	80	7.0	0.444	0.500	22.8	11.4	..	1
	24	192	6.5	85	7.0	0.492	0.443	27.4	12.1	..	1
14	22	176	6.	85.	7.0	0.489	0.483	25.1	12.1	..	1
	23	200	8.	100	7.0	0.417	0.500	28.6	14.3	..	1
15	24	192	6.5	90.	7.0	0.492	0.468	27.4	12.9	..	1
	25	200	7.5	105.	7.5	0.444	0.525	26.7	14.0	..	1
16	24	195	6.5	100.	7.0	0.500	0.513	27.8	14.3	..	1
	22	200	8.	105.	7.5	0.417	0.525	26.7	14.0	..	1
17	24	190	5.	100.	7.5	0.633	0.526	25.3	13.3	..	1
	22	201	7.5	112.5	7.0	0.447	0.560	28.7	16.1	..	1
18	23	184	6.0	100.	7.5	0.511	0.543	24.5	13.3	..	1
	24	210	8.	105.	7.5	0.437	0.500	28.0	14.0	..	1
19	23	185	5.5	100	7.0	0.561	0.540	26.4	14.3	..	1
	23	202	6.	90	8.0	0.561	0.445	25.2	11.3	..	1
20	22	200	7.	105	7.5	0.476	0.523	26.7	14.0	..	1
	25	200	8.	102.5	7.5	0.417	0.512	26.7	13.7	..	1
21	23	200	7.	115.0	6.5	0.476	0.575	30.5	17.7	..	1
	22	180	7.5	100.	7.0	0.400	0.555	25.7	14.3	..	1
22	23	200	6.5	110.	7.5	0.513	0.550	26.7	14.7	..	1
	24	200	5.	100.	7.5	0.667	0.500	26.7	13.3	..	1
23	23	200	4.5	100.	7.0	0.741	0.500	28.6	14.3	..	1
	26	208	6.5	90	7.0	0.533	0.433	29.7	12.9	..	1
24	23	200	5.0	120.	7.5	0.667	0.600	26.7	16.0	..	1
	23	201	6.0	100.	8.5	0.558	0.497	23.6	11.8	..	1
25	23	195	6.0	100	6.0	0.542	0.513	32.5	16.7	..	1
	24	192	4.5	90.	8.0	0.711	0.468	24.0	11.3	..	1
26	23	200	5.5	90.	7.0	0.606	0.450	28.6	12.9	..	1
	22	196	6.5	100.	7.0	0.503	0.510	28.0	14.3	..	1
27	24	200	5.5	110	7.0	0.606	0.550	28.6	15.7	..	1
	24	190	8.0	125	7.0	0.396	0.658	27.1	17.9	..	1
28	24	190	5.5	112.5	7.0	0.576	0.592	27.1	16.1	..	1
	25	190	7.	110.0	6.5	0.452	0.579	29.2	16.9	..	1
29	27	205	6.	110.0	7.0	0.569	0.537	29.3	15.7	..	1
	27	200	8.	137.5	7.0	0.417	0.657	28.6	19.6	..	1
30	29	210	6.5	125.0	7.0	0.533	0.595	30.0	17.9	..	1
	27	227	6.0	145.0	7.0	0.631	0.595	32.4	19.3	..	1
31	27	195	6.0	112.5	4.5	0.542	0.577	43.3	25.0	..	1
Totals		11,323	396.5	5,785.0	414.0						
Averages for the Month.....						0.476	0.511	27.35	13.97	2.10	

Averages for the Month..... 0.476 0.511 27.35 13.97 2.10

Average Rate Advanced for 59 Shifts=7.02 Ft.

TABLE I.—Data Compiled from Daily Reports.—(Continued.)

July, 1912. (2 drills.)	No of Holes.	Total Feet Drilled.	Actual Time Drill- ing. Hours.	Lb. Powder per Round.	Advance Feet.	Feet Drilled per Minute.	Lb. Powder per Ft. of Drill-Hole.	Feet of Drill-Hole per Foot Ad- vanced.	Lb. Powder per Foot Advanced.	Average Time of Drilling One Ft. in Minutes.	No Rounds.
1	34	210	5.5	61	6.	0.636	0 290	35.	10.2	...	1
	61	366	11.	189.	17.	0.555	0.516	21.5	11.1	...	3
	47	268	11.5	178	17.	0.388	0 664	15 8	10.5	...	3
2	21	126	5.	63.	6	0 420	0.500	21.0	10.5	...	1
	48	293	12.	104.	12.	0.407	0 355	24 4	8.7	...	2
3	24	144	6	56	6.	0.400	0 389	24 0	9.3	...	1
4
5	7	49	4.	71.	5.	0.204	1.450	9.8	14 2	1
6
7	22	132	6.5	65.	5.	0 338	0.492	26 4	13.0	1
	43	232	13	126.	12	0 323	0 500	21.0	10 5	2
8	25	150	5.5	69.	6	0.453	0.400	25 0	10.0	1
	43	252	12	119.	11	0 350	0 472	22 9	10.8	2
9	37	208	12.	113	11	0 289	0.546	18.9	10.3	2
	18	108	6.	50.	5.5	0 300	0.463	19 6	9.1	1
10	19	114	6.	112.5	11	0 816	0 987	10.4	10.2	2
	32	186	12.5	106.5	11 5	0 248	0 572	16 2	9.3	2
11	30	185	8.	114	11 0	0.385	0.616	16 8	10.4	2
	42	312	10 5	115	12.	0.495	0 369	26 0	9.6	2
12	33	198	6.	59.	6.	0 550	0 298	33.0	9.8	1
	19	101	6.	50	5 5	0.280	0.495	18 4	9.1	1
13	39	234	11	159.	17	0 355	0.680	13.8	9.4	3
	47	282	10.	185.	12.	0.470	0.656	23.5	15.4	2
14	26	156	6	70.	6.	0 433	0.449	26 0	11.7	1
	41	257	10 5	189.5	12.	0.408	0.733	21.4	15.8	2
15	25	150	4.	92.5	12.	0.625	0.617	12.5	7.7	2
	54	331	9.5	120.	13.	0.581	0 362	25.5	9.2	2
16	23	138	6	101.	13.	0.383	0.732	10.6	7.8	2
	37	220	7.5	107.5	12.	0.489	0.490	18.3	9.0	2
17	27	151	6.	97.5	11.	0 419	0.645	13.7	8.9	2
	52	307	10.5	107 5	12.5	0.487	0.350	24.6	8.6	2
18	19	135	6	102.	13.	0.375	0.755	10.4	7.8	2
19	3.0
	48	284	9.5	88.5	12	0.498	0.311	23.7	7.4	2
20	23	141	5.5	95.	11 5	0.427	0.674	12.3	8.3	2
	35	266	8.5	91.	12.5	0.404	0.481	16 5	7.3	2
21	22	132	6.	45	6.	0.367	0.341	22.0	7 5	1
	39	231	10.5	88.	11.5	0.367	0.381	20.1	7.7	2
22	17	102	5.5	88 5	12.0	0.309	0.868	8.5	7.4	2
	40	224	11.5	91.	11.	0.325	0 406	20.4	8.3	2
23	17	102	6.5	110.	10.	0.262	1.080	10.2	11.0	2
	35	200	11.5	115.	11.	0 290	0.575	18.2	10.5	2
24	17	100	6.	100.	11.5	0.278	1.000	8.7	8.7	2
	42	259	10.	107	11.	0.432	0 413	23.5	9.7	2
25	23	138	6.	65	10.5	0.333	0 471	13.1	2
	22	141	5 5	125.	6.	0.427	0.886	23.5	11.5	1
26	38	228	9.5	110.	12.5	0.400	0.483	18.2	8.8	2
	54	298	10 5	110.	11.	0.457	0.369	27.1	10.0	2
27	27	162	6.	111.	10.	0.450	0.685	16.2	11.1	2
	52	368	11.	128 5	12.	0.557	0.344	30.7	10.5	2
28	23	138	6.	138.	11.	0.333	1.000	12.5	12 5	2
	22	130	6.	70.	6.	0.361	0.538	21.7	11.7	1
29	40	240	11.	119.	12.	0.364	0.496	20.	9 8	2
	34	219	11.5	103	12.5	0.317	0 470	17.5	8.2	2
30	16	112	5.	91.5	12.5	0.373	0.817	9.0	7.3	2
	41	226	12.	106.	9.	0.314	0.447	25.1	11.8	2
31	12	84	2.	0.700
Totals		10,480	439.0	5,448.5	555.0						
Averages for the Month						0.398	0.520	18.9	9.8	2.51	

Average Rate Advanced for 80 Shifts—6.9 Ft.

TABLE II.—*Data Compiled from Daily Reports.*

SUMMARY.

Month and Year (Using 2 Drills).		No of Drill-Holes	Total Feet Drilled	Actual Time Drilling (Hours)	Pounds Powder Per Month.	Advance Gained (Feet)	Feet Drilled Per Minute	Pounds Powder Per Foot of Drill-Hole.	Feet of Drill-Hole Per Foot Advanced.	Pounds of Powder Per Foot Advanced.	Average Time of Drilling One Foot in Minutes.	Number of Shifts.	Average Rate Advanced Per Shift (Feet.)
June, 1911.	Totals	5,901 0	185.5	2,867.0	218.5	Total	1.47	advance	form.	380 ft.	33.
July, 1911.	Averages	11,323 0	396.5	5,785 0	414.0	0.530	0.486	27 0	13.1	1.89	59.	6.62
Aug., 1911.	Totals	11,059 0	427 5	6,935 0	403.0	0.476	0.511	27.35	13.97	2.10	7.02
Sept., 1911.	Averages	7,850.0	345 0	4,182 5	280.0	0.431	0.627	27 2	17.2	2.32	6.5
Oct., 1911.	Totals	8,703.0	475.5	6,043.5	351.5	0.379	0.533	28 0	14.9	2.64	5.83
Nov., 1911.	Averages	9,676.0	444.5	6,154.5	393.5	0.805	0.694	21.9	17 2	3.28	5.76
Dec., 1911.	Totals	5,037 0	280.0	2,408.0	282.0	0.363	0.636	24 3	15.4	2.76	6.64
Jan., 1912.	Averages	9,673.0	396.0	5,186.0	461.5	0.289	0.478	17 9	8.6	3 33	6.13
Feb., 1912.	Totals	9,176 0	329 5	4,559 3	402 5	0.407	0.536	20 9	11.2	2 45	7.44
March, 1912.	Averages	10,539 0	365.0	5,656 5	472.0	0.464	0.497	22 8	11 3	2.15	7.6
April, 1912.	Totals	1,210 5	51 5	326.5	40.0	0.484	0.537	22.4	12.0	2.08	8.4
May, 1912.	Averages	6,069.0	310.	3,011.2	205.0	Plus	62 ft.	cross	cut.
June, 1912.	Totals	11,888.0	442 0	6,768.0	488.0	0.326	0.496	29 6	14.7	3.06	8.3
July, 1912.	Averages	10,480 0	439.0	5,448.5	555.0	0.448	0.569	24 4	13.9	2.23	8.9
Aug., 1912.	Totals	8,049.0	311.5	3,201.5	421 0	0.398	0.520	18.9	9.8	2.51	6.9
Sept., 1912.	Averages	7,296 0	299.0	3,163.5	390.0	0.431	0.398	19 1	7 6	2.32	6.1
Oct., 1912.	Totals	3,302.0	125.5	894.0	185.5	0.407	0.434	18 7	8.1	2.45	5.7
Nov., 1912.	Averages	0.439	0.271	17 8	4.8	2.28	5.0
Grand Total	Totals	137,231 5	5,623.5	72,590.5	5,963.0	0.407	0.529	23.0	12.17	2.46	939.5	6.35
Grand Total	Length of Tunnel	6,235.5

The other lines on the chart, "Pounds of Powder per Foot Advanced," "Advance Gained," and "Pounds of Powder per Foot of Drill-Hole," were made, not only for our own study and information, but quite as much to encourage the men to be economical in the use of powder, and incite them to rivalry in the "Advance Gained per Shift." We have introduced isolated charts for two typical months only, because the continuous chart from the beginning of the tunnel to the end is about 9 ft. long and consequently seemed too long for convenient publication. (See Figs. 5 and 6.)

At first we operated on the basis of two 8-hr. shifts per day. Each shift drilled one full round with the view of "pulling"

8 ft. While following this plan our best month was July, 1911, in which our progress was 414 ft. We have taken this month, therefore, as typical of the most successful operation of the method. It will be seen that the maximum progress in any day was 15.5 ft.; and it is safe to conclude that more than 16 ft. was not possible.

With the view of devising a method which would at least render possible a greater progress per two shifts, we studied the problem carefully, on the basis of the data already obtained (such as the average feet drilled per minute, etc.), and came to the conclusion that two drillers and helpers, in good drilling-ground, could put in two rounds of from 4.5 to 5.5 ft. in one 8-hr. shift, which would make it possible under favorable conditions to advance from 18 to 22 ft. per day. The work of mucking under this method would be very much facilitated, because fully 25 per cent. more muck could be landed on the muck-plates, and, if necessary, a third shift of muckers could be used.

It is one thing to lay out a carefully worked out plan, but it may be very difficult to inaugurate the plan under favorable auspices. A good plan may easily fail by not having the enthusiastic support of the men essential to its success.

The following statement of the progress per month shows that September, October, November, and December were comparatively poor months. This was in part due to very difficult ground, containing a large amount of the jasper before mentioned. Nevertheless, in our opinion, a much better record could have been made had the plan of short rounds been adopted during this time.

Progress Per Month.

Month.	Year.		Feet
May,	1911	Last 24 days of month.....	111
June,	1911	Working two shifts.....	380
July,	1911	Working two shifts.....	414
August,	1911	Working two shifts.....	403
September,	1911	Working two shifts.....	280
October,	1911	Working two shifts.....	351.5
November,	1911	Working two shifts.....	393.5
December,	1911	Working two shifts.....	282
January,	1912	461.5
February,	1912	402.5

(Three days in this month lost on account of
accident to man.)

March,	1912	472
		Struck big flow of water on March 29, stop- ping all progress.)	
April,	1912	40
		(Practically had to stop work the whole of this month and part of May, on account of great flow of water and bad ground.)	
May,	1912	205
June,	1912	488
July,	1912	555
August,	1912	421
September,	1912	390
		(Much timbering required.)	
October,	1912	185.50
		(Much timbering required.)	
Total length of tunnel.		6,235.5

The tunnel reached the vein on Oct. 23 at a point 4 ft. from our calculation.

There was no favorable opportunity for trying out the plan of shorter rounds until Dec. 22, 1911. On that date we advanced 21 ft.; and, after this demonstration of the possibilities of the plan, there was no further trouble in putting it in force.

In applying this method, we found that the drill-men on an 8-hr. shift had to work overtime on an average of perhaps 1 hr., for which they were paid an extra amount. The drill-men and their helpers were glad to get this overtime. We found that with this method, in fairly good ground, two 8-hr. shifts of drill-men could pull as much ground as three 8-hr. shifts of muckers could handle. The method produced very efficient results all along the line. But we were not long permitted to derive benefit from it. Towards the end of March, we were notified by our attorney that a late decision of the Colorado courts made the working of men longer than 8 hr. (even though desired by the men) an offense punishable by fine and imprisonment. This had very much the appearance of putting a premium on inefficiency, for we were forced to employ two extra drill-men and helpers for 8 hr. in order to do 2 hr. work. This is the law in a State which, during the past few years, has been loud in its lamentations over the decline of mining. Under this mandatory condition, we could gain little, if any, efficiency and economy over our original method.

Towards the latter part of March, as noted in the foregoing statement of progress, a large flow of water was encountered, which almost stopped our progress for about six weeks.

This sudden influx of water was interesting, but very expensive. About Mar. 25 we struck the zone of faulting mentioned in the early part of this paper and shown on Fig. 2. In this zone the rock is full of slips and very much ground up. The tunnel was perfectly dry for a distance of about 40 ft. in this formation. On Mar. 29, a round of holes was put in, the bottom of each hole being perfectly dry. When this round was fired a flood of water came with it. The ground-up material of this formation seemed to disintegrate in contact with water and filled the entire face of the tunnel. It was roughly estimated that the volume of water amounted to more than 1,000 gal. per min. After trying in vain to catch up the ground at the face, the Superintendent came back about 50 ft. in the tunnel, and started a cross-cut at an angle to encounter the water some distance from the face, and thus divert the flow, or most of it, from the main tunnel. This plan succeeded; enough water was diverted to enable us to proceed with the driving of the main tunnel, although for many weeks it was only with the greatest effort that any progress whatever was made.

There is an interesting feature in connection with this matter, outside of the difficulties in the prosecution of the work. As we understand it, the zone of faulting constituted in effect a water-tight dam. Penetrating this dam with the tunnel, a point was finally reached where the pressure of water broke down the barrier remaining. That there was an actual hydraulic pressure back of this zone, seems to be clearly indicated by the fact that after several weeks of flow the force of the incoming water grew very rapidly less, and within a short time the flow became quite negligible.

No serious difficulty was encountered during the construction of the remainder of the tunnel, but progress during the latter part of August and during September and October, 1912, was much impeded by reason of the timbering required.

The maximum advance for any one month was 555 ft., made in July, 1912. However, we were forced, as already remarked, to use three full shifts of drillers as well as muckers during that month, and so do not consider that it equaled in economy and efficiency either January, February, or March; during which months two shifts of drillers were used.

We were much in error in our original estimate of the amount of timbering that would be required. All told, it was necessary

to timber 1,618 ft., most of which required full sets, instead of only 500 ft., as we had expected. Moreover, much of the timbering in August, September and October, 1912, had to be done so close to the face as to hinder seriously, and at times entirely stop, the progress.

Monthly Cost-Sheets and General Summary of Costs.

Each month, as the work progressed, we had two forms filled out, as shown in Figs. 7 and 8.

Form No. 1, Fig. 7, gives the detailed costs; Form No. 2, Fig. 8, the costs chargeable to the permanent plant. There were two reasons for keeping the record in this way. In the first place, bills for the various items under "Permanent Plant" came in very irregularly, and would have greatly delayed making out the monthly cost-sheet. In the second place, it was convenient in the end to have the various items of the "Permanent Plant" kept separate, so that the apparatus and material could be charged back to the general mine-equipment at a proper figure. We have charged back in this way the following items at 50 per cent. of their cost:

Drills and fixtures.....	\$647.01
Drill-sharpener and fixtures ..	1,031.17
Engine and blower	800.46
Air-pipe for blower and fixtures..	3,254.04
Compressor air-pipe.....	1,543.91
Drill-steel	916 48
Steel rails, bolts and spikes.....	1 849.78
Tram-cars.....	2,576.69
Mules and harness.....	756.24
Steel plate for muck	84.44
Water-pipe.....	391.52
Total.....	<u>\$13,851.74</u>

Taking 50 per cent. of the \$13,851.74 above given and which represents that part of the permanent plant to be salvaged, and dividing by 6,235.5, the length of the tunnel, we have \$1.111 to take from the total cost of \$20.986 per ft. of the tunnel, making the actual cost per ft. \$19.875.

Our original estimate on the cost of driving this tunnel was \$21.12 per ft. After deciding to construct the tunnel, we prepared plans and specifications, and advertised for bids. The lowest responsible bid received was \$26.50 per ft., and this did not include the timbering.

RAWLEY MINING COMPANY.

No. 12 Tunnel.

Monthly Report.

<i>Underground</i>	Month ending 19
Drilling and Firing—Labor	\$
Powder, Fuse and Caps	
Supplies and Repairs	
Mucking—Labor	
Supplies	
Tramming—Labor	
Supplies	
Track and Pipe—Labor	
Supplies	
Timbering—Labor	
Supplies	
General—Labor	
Supplies	
Supplies (paid for but not yet used)	
<i>Surface—</i>	
Power Plant—Labor	
Supplies and repairs	
Fuel	
Blacksmith—Labor	
Supplies	
General—Labor	
Supplies	
Supplies (paid for but not yet used)	
<i>General Expenses—</i>	
Salaries	
Office, traveling, &c.	
Freight, haulage and express, not otherwise distributed	
<i>Permanent Plant—</i>	
Previously reported	
Increased this month by	
Labor	
Supplies	
<i>Boarding House—</i>	
Disbursements—	
Labor	\$
Supplies	
Credits	
<i>Results—</i>	
Advance previously reported	
Advance for this month	
Total cash previously received	
" " received this month	
Total cash previously disbursed	
" " disbursed this month	
<i>Remarks—</i>	

Signed,

FIG. 7.—FORM OF DETAILED MONTHLY COSTS.

PERMANENT PLANT

For

No. 12 Tunnel.

Complete to 19

All labor.	\$4,300.06
Drills and fixtures	647.01
Drill-sharpener and fixtures	1,031.17
Engine and blower	800.46
Air-pipe for blower and fixtures	3,254.04
Compressor air-pipe	1,543.91
Drill-steel	916.48
Buildings and building supplies	933.45
Steel rails, bolts and spikes	1,849.78
Tram-cars	2,576.69
Mules and harness	756.24
Steel plate for muck	84.44
Telephone line	77.93
Freight, express, &c.	392.24
Compressor and boilers	335.77
Water-pipe	391.52
Miscellaneous	305 01
Total	<u>\$20,196.20</u>

FIG. 8.—MONTHLY FORM WITH FINAL CUMULATIVE COSTS.

From the monthly cost-sheets given above we have made up the complete and cumulative statement of costs which follows. The arrangement of this statement of costs can be most clearly understood by an example. Taking the month of June, 1911, it will be seen that "This Month's Cost" for "Drilling and firing" is \$2,160.49. This figure is the sum of the "Labor," "Powder, fuse and caps" and "Supplies and repairs," set immediately below the \$2,160.49. Under "Total Cost to Date" the \$2,921.88 is the sum of the \$2,160.49 and \$761.39, the "Total Cost to Date" of May. The "Cost per Foot Advanced to Date," \$5,951, is obtained by dividing the "Total Cost to Date," \$2,921.88, by 491 ft., the "Total advance." The figures for "Mucking," "Tramming," etc., are arranged in a similar manner.

	May, 1911			June, 1911		
	This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date	This Month's Cost	Total Cost to Date.	Cost Per Foot Advanced to Date.
UNDERGROUND.						
Drilling and firing.....	\$761.39	\$761.39	\$6.859	\$2,160.49	\$2,921.88	\$5.951
Labor.....	644.00			1,098.80		
Powder, fuse and caps.	117.39			1,013.13		
Supplies and repairs.....				48.56		
Mucking.....	169.87	169.87	1.530	701.29	871.16	1.774
Labor.....	153.00			689.13		
Supplies.....	16.87			12.16		
Tramming.....	58.50	58.50	0.527	100.31	158.81	0.323
Labor.....	58.50			100.31		
Supplies.....						
Track and Pipe.....				87.70	87.70	0.179
Labor.....				78.50		
Supplies.....				9.20		
Timbering.....	34.48	34.48	0.311	12.00	46.48	0.095
Labor.....	21.43			12.00		
Supplies.....	13.05					
General.....	173.71	173.71	1.565	415.78	589.49	1.200
Labor.....	114.00			341.88		
Supplies.....	59.71			73.90		
SURFACE.						
Power plant.....	153.87	153.87	1.386	863.67	1,017.54	2.072
Labor.....	94.50			373.75		
Supplies.....	15.37			158.72		
Fuel.....	44.00			331.20		
Blacksmithing.....	96.05	96.05	0.865	259.39	355.44	0.724
Labor.....	80.56			197.46		
Supplies.....	15.49			61.93		
General.....	273.67	273.67	2.466	212.12	485.79	0.989
Labor.....	234.88			166.97		
Supplies.....	39.29			45.15		
GENERAL.....	516.80	2,829.52	25.491	491.43	3,320.95	6.763
Salaries.....	370.00	1,520.00		390.00		
Office, traveling, &c.....	146.80	1,309.52		101.43		
FREIGHT, haulage, &c., not otherwise distributed.....	4.02	4.02	0.036		4.02	0.008
PERMANENT PLANT.....	3,294.02	6,553.92	59.044	3,577.03	10,230.95	20.837
Labor.....	1,137.91			260.85		
Supplies.....	2,156.11			3,316.18		
BOARDING HOUSE.....	657.68			558.02		
Labor.....	110.58			126.50		
Supplies.....	547.10			431.52		
Credits.....	226.45			599.35		
Balance, + or -.....	-431.23	-431.23	-3.883	+41.33	-389.90	-0.794
Advance previously made.....	00.			111.00		
Advance made this month.....	111.00			380.00		
Total advance.....				491.00		
Total.....			103.963			41.709

July, 1911.			August, 1911			September, 1911		
This Month's Cost	Total Cost to Date.	Cost Per Foot Advanced to Date	This Month's Cost.	Total Cost to Date	Cost Per Foot Advanced to Date	This Month's Cost	Total Cost to Date.	Cost Per Foot Advanced to Date
\$2,347.41	\$5,269.29	\$5.822	\$2,497.19	\$7,766.48	\$5.938	\$1,937.35	\$9,703.83	\$6.111
1,196.18			1,232.12			967.07		
973.96			1,120.15			615.96		
177.27			144.92			354.32		
769.33	1,640.49	1.813	680.24	2,320.73	1.774	582.98	2,903.71	1.829
749.99			674.60			576.62		
19.34			5.64			6.36		
262.53	421.34	0.455	280.48	701.82	0.537	241.76	943.58	0.594
201.25			214.38			208.51		
61.28			66.10			33.25		
139.47	227.17	0.251	128.31	355.48	0.272	110.67	466.15	0.294
139.47			128.31			110.67		
121.47	167.95	0.186	27.74	195.69	0.150	369.41	565.10	0.356
36.47			25.56			267.66		
85.00			2.18			101.75		
409.82	999.31	1.104	439.98	1,439.29	1.100	419.40	1,858.69	1.170
341.00			366.00			360.00		
68.82			73.98			59.40		
916.52	1,934.06	2.137	850.95	2,785.01	2.129	865.23	3,650.24	2.296
404.94			387.50			392.88		
51.27			3.83			30.07		
460.31			459.62			442.78		
169.47	524.91	0.580	317.39	842.30	0.644	213.14	1,055.44	0.665
152.07			215.56			201.34		
17.40			101.83			11.80		
277.70	763.49	0.844	276.83	1,040.32	0.795	175.83	1,216.15	0.766
238.43			204.70			160.69		
39.27			72.13			15.14		
438.50	3,759.45	4.154	425.94	4,185.39	3.200	414.80	4,599.69	2.896
390.00			390.00			390.00		
48.50			35.94			24.30		
	4.02	0.004	11.05	15.07	0.011	3.76	18.83	0.012
327.75	10,558.70	11.667	389.35	10,948.05	8.370	581.20	11,529.25	7.260
155.63			37.25			24.92		
12.12			352.10			556.28		
580.16			586.90			593.74		
124.50			126.00			127.30		
455.66			460.90			466.44		
614.05			620.25			542.91		
+ 33.89	-356.01	-0.393	+ 33.35	-322.66	-0.247	-50.83	-373.49	0.235
491.00			905.			1,308.		
414.00			403.			280.		
905.00			1,308.			1,588.		
		\$29.420			\$25.167			\$24.484

	October, 1911.			November, 1911.		
	This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.	This Month's Cost	Total Cost to Date.	Cost Per Foot Advanced to Date.
UNDERGROUND.	\$	\$		\$		
Drilling and firing	\$2,377.10	12,080.93	\$6.229	\$2,286.12	14,367.05	\$6.158
Labor	1,222.39			1,184.23		
Powder, fuse and caps..	940.47			997.81		
Supplies and repairs....	214.24			104.08		
Mucking	649.72	3,553.43	1.832	707.95	4,261.38	1.822
Labor	648.13			700.20		
Supplies	1.59			7.75		
Tramming	260.50	1,204.08	0.621	268.13	1,472.21	0.631
Labor	214.70			219.17		
Supplies	45.80			48.96		
Track and Pipe	129.54	595.69	0.307	124.42	720.11	0.309
Labor	129.54			114.41		
Supplies				10.01		
Timbering	49.50	614.60	0.317	19.46	634.06	0.272
Labor	42.00			3.50		
Supplies	7.50			15.96		
General	460.97	2,319.66	1.196	469.81	2,789.47	1.196
Labor	381.00			370.00		
Supplies	79.97			99.81		
SURFACE.						
Power plant	860.71	4,510.95	2.326	888.74	5,399.69	2.314
Labor	387.50			381.18		
Supplies	55.11			75.28		
Fuel	418.10			432.28		
Blacksmithing.. ..	445.76	1,501.20	0.774	298.96	1,800.16	0.780
Labor	308.83			277.98		
Supplies	136.93			20.98		
General	189.97	1,406.12	0.727	287.40	1,693.52	0.726
Labor	115.65			187.85		
Supplies	74.32			99.55		
GENERAL	439.50	5,039.19	2.598	412.55	5,451.74	2.337
Salaries	390.00			390.00		
Office, traveling, &c.....	49.50			22.55		
FREIGHT, haulage, &c., not otherwise distributed		18.83	0.010		18.83	0.008
PERMANENT PLANT	630.94	12,160.19	6.270	567.16	12,727.35	5.455
Labor	100.34			63.47		
Supplies	530.60			503.69		
BOARDING HOUSE	528.21			713.43		
Labor	127.00			130.00		
Supplies	401.21			583.43		
Credits	585.60			592.15		
Balance, + or -	+ 57.39	- 316.10	- 0.163	- 121.28	- 437.38	- 0.188
Advance previously made.....	1,588.			1,939.5		
Advance made this month.....	351.5			393.5		
Total advance	1,939.5			2,333.00		
Total			23.370			22.196

December, 1911.			January, 1912.			February, 1912.		
This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.	This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.	This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.
\$1,536.56	\$15,903.61	\$6.082	\$2,136.16	\$18,039.77	\$5.864	\$1,757.95	\$19,797.72	\$5.691
839.13			1,204.75			889.60		
379.40			747.64			716.28		
318.03			183.77			152.07		
655.69	4,917.07	1.880	949.41	5,866.48	1.907	880.84	6,747.32	1.940
647.24			930.79			873.59		
8.45			18.62			7.25		
256.14	1,728.35	0.661	390.25	2,118.60	0.689	529.47	2,648.07	0.761
231.04			361.10			457.76		
25.10			29.15			71.71		
158.99	879.10	0.336	186.14	1,065.24	0.346	146.73	1,211.97	0.348
158.99			186.14			146.73		
275.51	909.57	0.348	318.02	1,227.59	0.399	723.25	1,950.84	0.561
209.22			154.75			506.65		
66.29			163.27			216.60		
439.25	3,228.72	1.235	471.11	3,699.83	1.203	473.71	4,173.54	1.200
371.00			381.00			389.00		
68.25			90.11			84.71		
868.07	6,267.76	2.397	916.29	7,184.05	2.335	883.60	8,067.65	2.319
399.94			387.50			366.60		
63.13			78.96			54.15		
405.00			449.83			462.85		
255.14	2,055.30	0.786	346.91	2,402.21	0.781	284.15	2,686.36	0.772
209.81			284.33			242.63		
45.33			62.58			41.52		
251.19	1,944.71	0.744	331.06	2,275.77	0.740	270.78	2,546.55	0.732
173.63			247.66			209.10		
77.56			83.40			61.68		
527.40	5,979.14	2.286	692.22	6,671.36	2.169	737.03	7,408.39	2.129
390.00			400.00			400.00		
137.40			292.22			337.03		
18.83		0.007	18.83		0.006	18.83		0.005
585.48	13,312.83	5.091	945.06	14,257.89	4.635	1,637.08	15,894.92	4.569
30.13			.88			36.01		
555.35			944.18			1,601.02		
397.05			500.61			800.05		
150.00			146.80			148.50		
247.05			353.81			651.55		
572.35			533.45			593.15		
+ 175.30	-262.08	-0.100	+ 32.84	-229.24	-0.075	-206.90	-436.14	-0.125
2,333.00			2,615.00			3,076.5		
282.00			461.5			402.5		
2,615.00			3,076.5			3,479.0		
		\$21.953			\$21.149			\$21.152

	March, 1912			April, 1912		
	This Month's Cost	Total Cost to Date	Cost Per Foot Advanced to Date.	This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.
UNDERGROUND.	\$			\$		
Drilling and firing.....	\$2,127.83	21,925.55	\$5.550	\$148.08	22,073.63	\$5.531
Labor.....	1,032.01			78.00		
Powder, fuse and caps..	1,064.82			50.83		
Supplies and repairs.....	31.00			19.25		
Mucking.....	1,189.81	7,937.13	2.009	244.65	8,181.78	2.050
Labor.....	1,174.38			224.77		
Supplies.....	15.43			19.88		
Tramming.....	716.05	3,364.12	0.851	356.83	3,720.95	0.942
Labor.....	629.55			289.59		
Supplies.....	86.50			67.24		
Track and Pipe.....	199.50	1,411.47	0.357	144.99	1,556.46	0.391
Labor.....	199.50			144.99		
Supplies.....						
Timbering.....	359.87	2,310.71	0.585	1,044.05	3,354.76	0.841
Labor.....	290.46			872.77		
Supplies.....	69.41			171.28		
General.....	633.27	4,806.81	1.214	513.67	5,320.48	1.333
Labor.....	493.50			470.00		
Supplies.....	139.77			43.67		
SURFACE						
Power plant.....	968.51	9,086.16	2.287	950.54	9,986.70	2.257
Labor.....	387.50			379.55		
Supplies.....	90.83			72.99		
Fuel.....	490.18			498.00		
Blacksmithing.....	409.93	3,096.29	0.784	43.19	3,139.48	0.787
Labor.....	283.98			33.33		
Supplies.....	125.95			9.86		
General.....	346.49	2,893.04	0.732	318.05	3,211.09	0.805
Labor.....	294.20			303.00		
Supplies.....	52.29			15.05		
GENERAL	576.45	7,984.84	2.021	464.75	8,449.59	2.117
Salaries.....	400.00			400.00		
Office, traveling, &c.....	176.45			64.75		
FREIGHT, haulage, &c., not otherwise distributed.....		18.83	0.005		18.83	0.005
PERMANENT PLANT.....	622.43	16,517.35	4.181	511.84	17,029.19	4.267
Labor.....	13.12			70.55		
Supplies.....	609.31			441.29		
BOARDING HOUSE.....	467.02			598.72		
Labor.....	133.00			138.40		
Supplies.....	334.02			460.32		
Credits.....	635.85			652.50		
Balance, + or -.....	+ 168.83	- 267.31	- 0.068	+ 53.78	- 213.53	- 0.053
Advance previously made.....	3,479.			3,951.		
Advance made this month.....	472.00			40.		
Total advance.....	3,951.00			3,991.		
Total.....			\$20,844			\$1,976

May, 1912.			June, 1912.			July, 1912		
This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.	This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.	This Month's Cost	Total Cost to Date.	Cost Per Foot Advanced to Date.
\$								
\$1,220.12	23,293.75	\$5.551	\$2,792.79	26,066.54	\$5.570	\$2,502.62	28,589.16	\$5.457
776.05	1,377.45	1,294.17
328.78	1,163.24	1,066.91
115.29	252.10	141.54
473.10	8,654.88	2.063	982.15	9,637.03	2.057	1,156.80	10,793.83	2.060
470.35	982.15	1,156.80
2.75
388.40	4,109.35	0.980	540.45	4,649.80	0.993	701.05	5,350.85	1.021
323.50	535.50	605.95
64.90	4.95	95.10
151.80	1,708.26	0.407	131.95	1,840.21	0.395	267.50	2,107.71	0.402
151.80	131.95	267.50
.....
579.70	3,934.46	0.938	120.78	4,055.24	0.866	440.60	4,495.84	0.858
454.45	98.25	376.48
125.25	22.53	64.12
620.80	5,941.28	1.411	656.65	6,597.93	1.409	625.62	7,223.55	1.379
493.00	490.00	501.00
127.80	166.65	124.62
.....
953.40	10,940.10	2.607	899.51	11,839.61	2.528	1,021.92	12,861.53	2.455
401.18	375.00	397.70
45.68	51.39	75.82
506.54	473.12	548.40
241.57	3,381.05	0.806	272.89	3,653.94	0.780	262.03	3,915.97	0.748
207.50	256.10	230.70
34.07	16.79	31.33
331.96	3,543.05	0.844	323.34	3,866.39	0.825	338.42	4,204.81	0.803
222.69	281.90	289.90
109.27	41.44	48.52
1,067.24	9,516.83	2.268	590.22	10,107.05	2.158	714.42	10,821.47	2.066
400.00	400.00	400.00
667.24	190.22	314.42
.....
.....	18.83	0.005	18.83	0.004	18.83	0.004
292.19	17,321.38	4.128	544.39	17,865.77	3.814	742.19	18,607.96	3.552
7.50	21.25	38.50
284.69	523.14	703.69
735.83	771.48	794.11
145.70	144.65	163.50
590.13	626.83	630.61
662.85	740.05	858.20
-72.98	-286.51	-0.068	-31.43	-317.94	-0.068	+64.09	-253.85	-0.049
3,991.	4,196.	4,684.
205.	488.	555.
4,196.	4,684.	5,239.
.....
.....	\$22.076	\$21.467	\$20.854

	August, 1912.			September, 1912.		
	This Month's Cost	Total Cost to Date	Cost Per Foot Advanced to Date.	This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date
UNDERGROUND.						
Drilling and firing	\$1,797.96	\$30,386.92	\$5.369	\$1,568.85	\$31,955.77	\$5.282
Labor	1,030.35	1,007.47
Powder, fuse and caps....	571.19	456.40
Supplies and repairs	196.22	104.98
Mucking	1,083.45	11,877.38	2.098	999.23	12,876.51	2.128
Labor	1,083.45	999.23
Supplies
Tramming	616.10	5,966.95	1.054	603.75	6,570.70	1.086
Labor	611.20	603.75
Supplies	4.90
Track and Pipe	258.55	2,366.26	0.418	147.25	2,513.51	0.416
Labor	258.55	147.25
Supplies
Timbering	1,035.42	5,531.26	0.977	731.63	6,262.89	1.035
Labor	823.60	498.93
Supplies	211.82	232.70
General	657.39	7,880.94	1.392	569.76	8,450.70	1.397
Labor	496.00	475.00
Supplies	161.39	94.76
SURFACE.						
Power plant	1,009.97	13,871.50	2.450	1,011.51	14,883.01	2.460
Labor	387.50	381.70
Supplies	42.12	61.53
Fuel	580.35	568.28
Blacksmithing	229.05	4,145.02	0.732	199.60	4,344.62	0.718
Labor	209.10	181.55
Supplies	19.95	18.05
General	343.53	4,548.34	0.804	330.38	4,878.72	0.806
Labor	306.50	283.52
Supplies	37.03	46.86
GENERAL	504.20	11,325.67	2.001	516.25	11,841.92	1.957
Salaries	400.00	400.00
Office, traveling, &c.	104.20	116.25
FREIGHT, haulage, &c., not otherwise distributed.	18.83	0.003	18.83	0.003
PERMANENT PLANT	480.23	19,088.19	3.373	510.72	19,598.91	3.240
Labor	19.90	13.50
Supplies	460.33	497.22
BOARDING HOUSE	633.36	682.54
Labor	159.70	155.00
Supplies	473.66	527.54
Credits	735.10	692.40
Balance, + or -	+ 101.74	- 152.11	- 0.027	+ 9.86	- 142.25	- 0.023
Advance previously made....	5,239.	5,660.
Advance made this month...	421.	390.
Total advance	5,660.	6,050.
Total	\$20.708	20.551

October, 1912		
This Month's Cost.	Total Cost to Date.	Cost Per Foot Advanced to Date.
\$795.88	\$32,751.65	\$5.252
530.40
181.80
83.68
582.70	13,459 31	2.157
582.70
.....
471.65	7,042.35	1.129
411.65
60.00
246.65	2,760.16	0.443
246.65
.....
1,110.82	7,373.71	1.182
724.15
386.67
543.92	8,994.62	1.443
444.00
99.92
.....
1,067.28	15,950.29	2.558
362.75
73.58
630.95
200.21	4,544.83	0.729
160.00
40.21
321.54	5,200.26	0.834
279.15
42.39
508.35	12,350.27	1.981
400.00
108.35
.....
.....	18.83	0.003
597.29	20,196.20	3.239
54.35
542.94
666.53
155.50
511.03
585.85
— 80.68	— 222.93	— 0.036
.....
6,050.
185.50
6,235.50
.....
.....	\$20.986
.....	1.111
.....
.....	\$19.875
.....	Less 50 per cent. of permanent plant.
.....	Net cost per foot.

It is to be noted in analyzing these figures that there are some seeming inconsistencies. For instance, the cost of mucking gradually increased throughout the course of the work. Theoretically there should be no difference in the cost of shoveling rock into a car whether it is 5,000 ft. from the portal or 500 ft. The explanation for this increase is threefold: first, the labor cannot be maintained in exact ratio to the progress (if three men can muck a maximum of 8 ft. per shift, four men must be used to muck 9 ft.); second, if the tunnel breaks large there is more muck per foot advanced, and the cost per foot increases; finally, the efficiency of the mucking crew decreased, in our opinion, for various reasons, as the work progressed.

Another apparent inconsistency is the gradually-increased cost of track- and pipe-work. This was very probably due to the fact that with every foot advanced there was 1 ft. more of track, pipe, and ditch to maintain and keep in repair.

We submit the foregoing discussion of the mining problem, together with the complete data regarding the driving of the tunnel, in the hope that it will have value to those who may be confronted in the future with somewhat similar situations. As regards the tunneling data, we have tried to make them complete, and consequently it is a record of unsatisfactory as well as satisfactory results. We trust it may serve both as a warning against uneconomical methods, and as an aid to securing greater efficiency in work of this character.

In conclusion, we wish to pay a deserved tribute to the men whose labors contributed in large measure to any success attained. No more fitting comment can be made than the statement that the work throughout was delayed less than 30 min. on account of trouble with the power-plant, and this statement simply illustrates the efficient service rendered in every department. To Will C. Russell, who had charge of this work, and to Charles E. Beckwith, his assistant, unstinted praise is due for their unflagging zeal, fidelity, and vigilance.

DISCUSSION.

W. L. SAUNDERS, New York, N. Y.:—Mr. Burns in his native Colorado modesty has said that he apologizes for things left out of this paper. I am much impressed by what he has put into the paper, as of all the papers that I have read on the sub-

ject, describing, as it does, a specific piece of work that has been accomplished, I do not recall one that gives so much information in such a practical way. Mr. Burns has also said that it is very hard to get men to put things down while the work is being driven. My experience is that it is very much harder to get the men in charge of work of this kind to bring them in in the form of a report before a body of engineers after the work is done. We have a great many cases where records are kept in much detail, but they are usually kept under cover, with the mistaken idea that the information gained is private information for the benefit of those who have done the work. Messrs. Simonds and Burns have opened up this case so that we can get the benefit of that experience which they have had, and so that we can learn something for the future.

The description contained in this paper, as I have said, is a very practical one. You will notice in going through it that the authors have given in detail a record of their work from the beginning to the end, and they even go so far as to publish an exact copy of the Rawley Mining Co.'s daily report. That daily report is something which those of us who are engaged in similar work might copy to great advantage, and having it before us we can check up our men while doing work and exact from them equal conditions, if it is possible to do so.

The dimensions of this tunnel are 7 ft. high by 8 ft. wide, with a length of a little over 6,000 ft., so you see we have here a typical mining tunnel, a little over one mile long, and with the usual dimensions of 7 by 8 ft. The best monthly progress, firing from 8- to 12-ft. rounds, was 414 ft., and the best monthly progress for the shallow rounds, from 4.5 to 5.5 ft., amounted to 555 ft. I want to call particular attention to that statement, and those of you who have read the paper will see that when they changed from a double shift to a triple shift, putting in shallow holes, they got better progress. The shallow hole idea is the European idea, it is the Alpine tunnel idea, it is the idea by which engineers have carried through those great enterprises of piercing the Alps, in the case of the Simplon tunnel and the Arlberg and the Loetschberg tunnels. These recent tunnels which have been put through the Alps have all been driven on the shallow hole plan; in other words, a hole is put in about a meter or a little more than a meter in depth, and of

rather large diameter, as compared with the American system, which is, a hole as large as you can reasonably get it, and that means, by the time it is in to the bottom, say 10, 12, or 14 ft., it is just a little larger than the diameter of the steel, the result being that you are putting the explosive at the bottom in a section of hole of small diameter, which is the very reverse of what is wanted in order to produce the most effective results. It is much better—and this is a point that I have emphasized before the Institute in papers in the past years—to put in a large number of shallow holes of large diameter and use a great deal of powder than to put in a small number of deep holes and try to save powder and try to be very particular about the exact direction that the hole must take. The European system simply peppers the head with holes, and better results are obtained by that method.

The total cost per foot of advance in the completed tunnel was \$19.87. That is a very fair figure, and less than the estimate of cost made before this tunnel was driven. The average total cost per cubic yard excavated was \$7.64. That may seem high to some of us, but if you go over the figures of labor cost in this paper you will see that the wages are higher. The people employed on this work evidently were compelled to pay a maximum rate of wages for this class of work, and, taking that into consideration, the figures are not high. The average number of feet of hole per drill per drill-hour was 12.2. The average inches per drill per minute was 2.5. The approximate yardage per foot of advance, allowing for 25 per cent. of overbreakage and swelling, was 2.6 yards, or a cost of \$7.64 per cubic yard. In all of this we must take into consideration the difficult rock which was encountered and the water troubles which they had; yet, comparing the progress of this tunnel with that of any of the recent tunnels of which we have records, the progress appears to be very favorable. The most recent of these records was made at the Arizona Copper Co.'s mines at Morenci, Ariz., the tunnel being 8 by 9 ft., where 780 ft. advance was made in 29 days, employing two C-110 drills mounted upon a cross-bar. We also have the record of the Laramie tunnel, in Colorado, which is similar to this one except that it has a ragged section. It is nothing but a hole, without any specified dimensions of outline, but it serves its

purpose very well as a drainage tunnel. Being ragged, it enabled progress to be made in excess of that given in this paper.

Now, the description of the process of driving tells us that this tunnel was driven by the system of mounting the drills on a horizontal cross-bar across the tunnel. Here we have a thoroughly modern system; the same, by the way, as was used at the Laramie tunnel. In the American system of tunnel-driving vertical columns with arms on them are usually employed. The European system of tunnel-driving is one in which horizontal bars mounted on carriages are employed. The system described in this paper is neither one nor the other, but the drillers used a horizontal bar without any carriage; in other words, they adopted the Alpine system of tunnel-driving, but did away with the carriage. Now, they were able to do away with the carriage because they used hammer drills instead of piston drills in the headings. The piston drill is a machine which reciprocates and which creates a great deal of shock on its mounting, necessitating a very heavy bar or column. The machine itself is heavy, and necessitates the use of either the carriage system or the vertical column with arms.

It appears that those who drove this tunnel used the hammer drill, which is lighter in weight and does not give the jarring and reciprocation, hence they were able to use a light bar, which a few men in the heading can handle to advantage. This light bar is carried on top of the muck pile after the blast and jacked in place close to the roof; the drills are put on and put in operation; the men work the drills standing on top of the muck, the muckers being at work at the same time in digging away the muck underneath them. By the time the muck is largely removed the drillers are about ready to let the bar down, and just about that time the muck has been taken away, so that if necessary the bar can be placed in a lower position in order to put in the lower round of holes, if these cannot be drilled from a single setting of the cross-bar. By such a system as this they were able to make a progress, I believe, of 26 ft. per day; in other words, a little more than 1 ft. an hour during all of the 24 hours in the day. That, of course, is not the record. The Alpine system has made about 30 ft. per day, and in the Laramie tunnel they made something like 24 ft. a day. Comparing progress of this character, however, we must

always take the character of the rock into consideration; also the length of the tunnel. The Alpine tunnels are longer tunnels, and, by adopting a certain system and carrying it out for a long period of time, it was possible to get the efficiency of the workers very high.

The engineers in this tunnel decided that better results could be obtained by drilling shallow holes and operating three shifts per day. This plan was put into effect in December, 1911, and, as I understand from reading the paper, it proved to be very efficient. The results, as given in the table printed in the paper, show that the average number of feet of hole per drill per drilling hour was 12.2. This is good, but not remarkable, as it involves only approximately 2.5 in. per min. Of course, the rock conditions govern this. On page 171 reference is made to the results obtained in the entire equipment of the tunnel.

This subject is one which is very active at the present time, because American engineers are adopting a system of tunnel-driving which is to a certain extent a cross between the European and the old American system, but which has, I think, a great many points of advantage over both of them.

THE CHAIRMAN:—May I ask, when the tunnel got to the vein did it satisfactorily drain out all the water, or did the water go down through the vein itself or through the false zone outside?

MR. BURNS:—We reached the vein on Oct. 23, but we did not have to wait to know that it would drain; the influx of water informed us of that. The upper levels drained in about 30 days.

THE CHAIRMAN:—There was an open connection right through the tunnel?

MR. BURNS:—Yes.

PROF. HENRY S. MUNROE, New York, N. Y.:—I hope the authors will supplement this paper with additional details regarding the method of ventilation employed. I note that they used a rather small pipe for so long a tunnel. From the scale of the drawing I should judge it was about 12 in. in diameter. It appears also that the heading was ventilated by exhaust in-

stead of by pressure, and that they used a Roots blower, indicating that a high water-gauge was employed.

As the use of auxiliary fans is common in mines, and as we have little information as to the working results, I hope that the authors can give us some idea as to the leakage in this long pipe; and as to the type of joint used, with the method of calking or wrapping, or the precautions taken to lessen leakage. It would be of interest to know the actual amount of air entering the pipe at the face of the tunnel as compared with that discharged by the fan; also, the water-gauge at the fan, the number of revolutions necessary, and the horse-power required.

It would be of interest also, as bearing on the relation between diameter and thickness of metal, to know whether the air pressure from heavy blasts tended to collapse the pipe, how near to the face the pipe was laid, and whether suction was efficient in removing the powder smoke quickly after a blast.

MR. BURNS:—I think I can furnish a good deal of the data that Professor Munroe asks for. The information he desires is among those things we did not give and is, perhaps, conspicuous by its absence from our paper. We got good results. To show the volume of air which our blower sucked out, I can say that in 20 min. our heading was clear of smoke, so that the men could go in and go to work. That was the practical thing that we were looking for, and possibly we overlooked making as careful records in regard to work of that character as we otherwise should have done. We got the results, and that was our chief object.

Now, this much is certain: our pipe work was done pretty well. I think if there had been any very serious leakage in our pipe it would have been discovered very soon. A leakage 1,000 ft. from the portal, when we were in 5,000 ft., would not have had to be very large in order to interfere seriously with sucking the smoke out rapidly and efficiently, and our men who repaired the pipes and took care of the track were constantly on the lookout for any leakage and immediately repaired it. A serious leak would become quite evident on account of the noise made; and further, our men carried their lights along the pipe, and a leak of any kind was very soon discovered.

At this time I wish to call attention to the cost data which we give on the track and pipe work: You will notice that the track and pipe expense shows a gradual increase from the first month to the end. At first I was puzzled to know why that should be, but I believe that it was because every foot of pipe and track completed became just that much more to keep in repair.

MESSRS. SIMONDS AND BURNS (communication to the Secretary *):—In response to certain suggestions by Prof. H. S. Munroe, we take pleasure in submitting the following details:

The ventilation pipe consisted of No. 18 U. S. standard gauge, riveted steel, coated with asphalt, 12 and 13 in. diameter, in lengths of about 20 ft. Slip joints with lugs were used, and tar paint was used on all joints. It was only occasionally necessary to use the lugs, in which event bailing wire was looped around the lugs and twisted till the necessary tension was produced. In case any leak developed in a joint a thick layer of tar paint was spread on a strip of canvas and bound tightly over the joint.

It was occasionally necessary to remove a length of the ventilation pipe either on account of damage from falling rock or because timbering became necessary. In order to provide for this contingency lengths of flanged pipe were put in every few hundred feet.

The reason for using two sizes of pipe was that, in shipping, the pipes could be "nested," and consequently not only save freight but somewhat lessen the liability to damage.

It is quite probable that for a permanent length of 6,000 ft. a ventilating pipe 12 in. in diameter would not be the efficient size to adopt, cost of pipe and cost of fuel properly considered, but it must be remembered that the distance was not 6,000 ft. but zero to 6,000 ft. Anticipating an extra demand on both engine and blower as the tunnel neared its maximum length, we provided in the beginning an engine 9 by 9 in., instead of the 8 by 8 in. engine more commonly supplied by the manufacturers for this size blower.

It may be of interest to discuss this problem of ventilation from an initial point of view—whether to ventilate by "pressure" or "exhaust."

* Received May 3, 1913.

So far as the question involves the operation of the engine and blower, either method could be adopted, but it is far different when the question of installing and operating the pipe is taken into account. The pipe when operating under exhaust can be installed much more rapidly because very little care is required in order to make tight joints. In case leaks develop at the joints the defect can be easily and quickly remedied; in fact, this is true of almost any damage that is likely to occur, such as the penetrating of the pipe by the sharp corner of a rock or the sharp point of a pick. No tendency in the pipe to collapse appeared at any time either from the exhaust pressure or from the effect of blasting. Practically no damage of any kind occurred to the pipe except as the result of falling rock, and this, considering the treacherous nature of much of the ground, was of infrequent occurrence.

Finally, considering the question from the standpoint of the results to be accomplished,—viz., the removal of the noxious gases in the least possible time,—the reasons seem to be very conclusively favorable to the suction method, at least after the face of the tunnel has reached a point about 1,000 ft. from the portal. Whichever method is employed, the pipe cannot safely be carried much closer to the face than 150 ft. In either case, also, just before blasting, the compressed air nozzle would be placed some 35 ft. from the face, pointing directly thereat, and the air turned on. This not only stirs up the gases at the face but quite rapidly forces them towards the ventilating pipe. In case of a suction pipe this action is accelerated, but in case of a pressure pipe the action would be retarded. Assume, for instance, that the tunnel is in 6,000 ft. and that the blower has a capacity of 8 cu. ft. per rev., or 2,400 cu. ft. per min. Under this condition it took rather less than 20 min. to suck practically all the noxious gases from the face. In 20 min., at the above rate, 48,000 cu. ft. would be removed, and this corresponds to the removal of air (expansion, contraction, and compressed air not considered) from 860 ft. of the tunnel. Comparing this with the pressure method under similar conditions, if the incoming air from the blower could immediately collect the noxious gases in a body and push them towards the portal it would take (expansion, contraction, and compressed air not considered) some 2.5 hr. to remove them from the tunnel. Of course in this latter case there would result a gradual mixing

of the air and gases, making the same breathable but probably not very healthy.

Following out suggestions made by Professor Munroe, we have had some measurements made of velocity of air and vacuum produced by the fan. The pipe was partly dismantled, but 4,650 ft., measured from the blower, was still in place. Making correction for error in the anemometer readings, the velocity of air obtained was approximately 3,000 ft. per min. at inlet and discharge, at 300 rev. per min. of the fan. The vacuum at the fan was 2.5 lb. From this data a rough calculation makes the theoretical velocity about 2,800 ft. per min. Water is discharged with the air at the outlet, making it impracticable to measure the velocity accurately, but the leakage is apparently negligible.

The comments by Mr. Saunders are interesting, and particularly gratifying to us in one respect—the facility with which he has transformed our data into other terms for the purpose of consideration and discussion. This was one special object which we had in view, viz., to give the data so completely and specifically that engineers could use the same for estimates and comparison free from doubt as to the inclusiveness of any particular item.

We are of the opinion that Mr. Saunders was either under a misapprehension in one respect, or inadvertently misstated himself. We refer to the statement “that when they changed from a double shift to a triple shift, putting in shallow holes, they got better progress.” Of course we naturally would have secured better progress with a triple shift, but what we particularly wished to emphasize was the fact that when we changed from a *double shift* using long holes to a *double shift* using shallow holes, but with *three shifts* of muckers, we secured better progress and greater efficiency. The maximum progress possible, as pointed out, with a double shift using long holes was 16 ft., while with a double shift using shallow holes 19 ft. and over was not at all unusual, as shown by the following record :

	Feet.		Feet.
Jan. 15.....	22.0	Mar. 3.....	19.5
Jan. 23.....	19.5	Mar. 5.....	21.5
Jan. 30.....	19.5	Mar. 6.....	21.5
Feb. 29.....	20.0	Mar. 18.....	22.0
Mar. 2.....	19.0	Mar. 24.....	19.0

Electric Power Installation at El Tigre, Sonora, Mexico.

BY JAMES W. MALCOLMSON, KANSAS CITY, MO.

(New York Meeting, February, 1913.)

THE Tigre Mining Co. of Mexico, owned by the Lucky Tiger Combination Gold Mining Co. of Kansas City, decided early in 1910 to enlarge its mill, which consisted of a concentrator milling 3,000 tons of ore per month. The enlarged plant was designed to crush and concentrate 6,000 of ore and to cyanide 7,500 tons per month. The extra tonnage cyanided was to be obtained from tailings which were stored below the old mill.

Power was supplied to the old mill from four boilers; a 150-h-p. Corliss engine and two Weber 60-h-p. gas-engines using producer-gas made from charcoal.

The total horse-power required, including 60 h-p. used for machine-drills in the mine, was approximately 275. Cord-wood cost \$4.32 per cord, 8 by 4 by 3 ft., and charcoal \$15 per ton. The fuel supply in the surrounding country was approaching exhaustion; more than 150 mules and 50 men were engaged in cutting and transporting this fuel, and the unsettled political conditions in Sonora were such that it would be practically impossible to keep an organization of this kind together. The price of fuel was steadily rising and at times it became almost impossible to get sufficient fuel to operate.

It was estimated that the enlarged plant, together with the mine, would require 750 h-p., as follows:

	Horse-Power.
Old mill, which was left unchanged,	180
New concentrator,	90
Cyanide-plant,	380
Mine and other requirements,	100
Total,	<u>750</u>

In a study of the best manner to obtain power the following alternative propositions were considered:

1. To continue using the local fuel supply by building tramways and cheapening transportation generally, thereby reaching a larger area of country. Power to be generated by means of wood-burning gas-producers and gas-engine plant.
2. To build a steam-power plant at Yzabal, the nearest railroad-point, 30 miles away, and transmit electric power to the mine, using Texas or California oil as fuel at \$1.70 (U. S. Cy.) per barrel on the assumption that the Mexican government would remove the import duties.
3. To build a dam on the Bavispe river, 10 miles from the mine, and install a hydro-electric plant and transmission-line.
4. To install a power-plant at Douglas, Ariz., and transmit electric power 65 miles to the mines.

An investigation of the available timber showed that it consisted of 80 per cent. of white pine and 20 per cent. of scrub oak. The supply was limited and much of the pine would be required during the next 10 years for mine timbering and general construction work.

It is estimated that 13,500 cords of wood, 8 by 4 by 3 ft., would be required, per annum, to produce 750 h-p. in a well-designed gas-engine plant, as follows :

	Horse-Power Hours per Annum	Consumption per H-P-hr.	Total Quantity per Annum.	Per Cord.	Total per Year
White dry oak, . .	1,600,000	Pounds. 3.3	Pounds. 5,280,000	Pounds. 2,020	Cords. 2,000
Dry pine	4,800,000	3.0	14,400,000	1,326	10,900
	6,400,000		19,680,000		13,500

Estimating these 13,500 cords at \$3.25
 per cord equals a total of \$43,875
 per annum. Add coke used in the bottom of the producers,
 1½ per cent. of the weight of the wood consumed, equals 150
 tons per annum at \$20 per ton, equals per annum, 3,000
 Total fuel cost (U. S. Cy.), \$46.875

A well-designed steam-plant at Yzabal with electric power transmission to El Tigre would consume 1.5 lb. Texas fuel-oil per horse-power delivered to the motors. Taking 750 h-p. for 365 days equals approximately :

6,400,000 h-p-hr. at 1.5 lb.,	9,600,000 lb. of oil
311 lb. of oil per barrel equals,	30,870 barrels of oil
\$1.70 per barrel at Yzabal gives total fuel-cost per annum	
(U. S. Cy.),	\$52,480
Excess cost over gas fuel at the mine	5,605

This difference would be wiped out by an increase of \$0.50 per cord in the cost of 13,500 cords per year. In fact, before the new power-plant was in operation the price of cord-wood had increased \$1 per cord and the cost under any circumstances would have steadily risen.

Even with wood at \$3.25 per cord the estimated saving was more than offset by the greater expense for labor, repairs, and supplies with gas-engines.

In a plant of this size the expense of gas-power would be nearly twice as much for repairs and supplies as for the corresponding items in a steam-power transmission-plant. The cash expenditures per annum would be greater at El Tigre than with a steam electric plant at Yzabal, even if there were no increase in the cost of wood.

Regarding first cost, it was estimated that gas-engine power is unsatisfactory unless the plant has one spare engine to allow for the periodical cleaning of each engine in turn.

It was estimated that a suitable plant of gas-producers, gas-engines and dynamos could only be installed at El Tigre, 30 miles from the railroad, for \$150,000, taking the European cost of engines. Engines made in the United States would cost more.

The first cost of a suitable steam-plant at Yzabal with electric power transmission to El Tigre was estimated at \$213,400, including economizers, as follows:

Power-plant at Yzabal erected complete,	\$133,000
Economizers (not considered important),	17,000
Transmission line to El Tigre, 30 miles, 30,000 volts,	30,000
Step-up and step-down transformers,	14,000
Margin on a preliminary estimate, 10 per cent.,	19,400
Total estimated cost,	\$213,400

The estimated cost of delivered horse-power per year using 750 h-p. was \$105.

Compared with power transmitted from a steam-plant at Yzabal, gas-engine power at El Tigre would be less reliable, and

more troublesome, and the proposition to install a local power-plant at the mine was, therefore, definitely rejected.

After an expert examination of a proposed dam on the Bav-ispe river, this proposal was also abandoned. The river is usually 150 ft. wide and 2 ft. deep, but at times it is 2,000 ft. wide and 24 ft. deep, and during dry seasons the river is not over 6 in. deep. If a dam and power-plant were built, except at enormous cost and many miles away, a duplicate steam-plant would also be necessary and the total estimated cost proved to be out of all proportion, when compared with the other projects.

Before finally deciding to build the plant at Yzabal, which is 45 miles south of Douglas, an investigation was made as to the possibility of securing power from that town. In Douglas, the power-plant of the Copper Queen Consolidated Mining Co. develops approximately 5,000 h-p. for the requirements of its smelter. An arrangement was made whereby this company agreed to install exhaust-turbine generators in its plant and supply electric power to the Tigre Mining Co. The installation was to be paid for by the Tigre Mining Co., and the power sold on a sliding-scale varying with the amount taken and the cost of fuel-oil in Douglas. In August, 1912, this amounted to 0.96 cent per kw-hr. at Douglas.

The installation of an exhaust-turbine plant at Douglas presented several advantages, although the cost of the machinery, together with the 65-mile transmission line to the mine, was considerably in excess of the Yzabal proposition with a 30-mile transmission.

The principal advantage, which led to a final decision to adopt the Douglas proposal, was the cost of power.

Taking 750 h-p., it was estimated that a plant at Yzabal would cost per annum \$79,000, or \$105 per horse-power delivered at El Tigre per year.

Power from Douglas delivered at the mine, it was estimated, would cost slightly more than \$70 per horse-power per year, or from \$54,000 to \$57,000 per annum, depending on whether one or two turbines were required.

In other words, the Douglas proposal showed a saving in cost of power of \$25,000 per year, and it was decided in July, 1910, to adopt it and build a line to the mine.

In June, 1911, power was delivered over the line to El Tigre,

the revolution in Mexico causing some delay in prosecuting the work.

The plant at the works of the Copper Queen Consolidated Mining Co. in Douglas consists of two 750-kw. exhaust-turbine generators which will operate with a 50 per cent. underload or overload without any very serious loss of efficiency. The Tigre Mining Co. receives the power at the bus-bars at a tension of 2,200 volts. This is stepped up to 44,000 volts by means of three General Electric transformers. At the mine the current is stepped down to 440 volts and distributed to the various circuits in the plant.

The transmission is peculiar on account of the small quantity of power being transmitted such a long distance.

The line is a 44,000-volt, 60-cycle, 3-phase line consisting of a single line of wooden poles carrying 3 conductors of No. 4 B. & S. gauge medium hard drawn copper with telephone wires below. The poles are spaced 200 ft. apart; and at the crossing of the Bavispe river the span is 1,600 feet.

The cost of the line from the low-tension side of its step-up transformer station at Douglas to the low-tension side of its step-down transformer station at El Tigre was \$161,121. Not including the transformer stations at each end the cost was very closely equal to \$2,000 per mile. The line, including the transformer stations, was built by Messrs. Sanderson & Porter, of New York.

The total cost of the exhaust-turbine generator-plant, including the steam-piping, etc., was \$71,894, the machinery being installed by the Copper Queen Consolidated Mining Co.

During the past year 6,000 tons of ore have been concentrated and 7,500 tons cyanided monthly at the Tigre mill. An average of 616 h-p. is distributed at the El Tigre switchboard at a cost of \$86 per horse-power per year; the cost at Douglas being 0.95 cent per kilowatt-hour.

The Hardinge Conical Mill.

BY H. W. HARDINGE, NEW YORK, N. Y.

(New York Meeting, February, 1913.)

NEARLY every mining and metallurgical engineer will recall his early experience and method of producing step- or stage-reduction in preparing ore-samples for assay, in which he employed ideal step- or stage-reduction simply because it was a self-evident fact that it would be easier to break coarse ore with a hammer than it would be to roll it back and forth under the muller, and after having reduced it to a size easy for a rubbing or bucking division he then placed it under the muller on the bucking-board and further reduced it in proportion to the physical energy he wished to expend, which was generally the minimum to produce results. After this step in our experience we seem to have ceased to consider stage-reduction as mechanically essential, probably because it did not apply to individual exertion. Moreover, we did not retain the mass under the muller until it was all reduced to pass the 80-mesh assay-office screen, but practiced further step-reduction by screening out the fine in order that it should not interfere with subsequent work, replacing the coarse for further reduction on the bucking-board.

It was many years after the bucking-board stage of metallurgical practice that we were brought to a full realization that there was a more economical method in applying the crushing forces usually employed in metallurgical work by taking advantage of this individual experience and applying it to mechanical ends. We have made some advance to this end in bringing out the conical ball-and-pebble mill—a device which fairly well automatically adjusts power to the results obtained, mechanically repeating the bucking-board experience. In the action of the conical mill, theory is evolved from practice, rather than practice from theory, as is commonly the case.

The conical mill, or what is more commonly known as the Hardinge mill,¹ is one of those inventions which, if not fathered by actual necessity, was at least induced by the desire to get better results from the energy expended in operating the older types of pebble-mills in my metallurgical work. The more common use of pebble-mills goes back some 10 or 12 years, while their first commercial introduction will date to about 1895 or 1896; at least, it was about that time that I first came across the cylindrical pebble-mill, and while I was not a theoretical convert upon sight, I soon became a practical or empirical convert upon trying out the device. In studying the action of the old style of cylindrical tube-mill, Fig. 1, I could not reconcile the fact that initial and final energies remained the same throughout the length of mill in spite of previous reduction of size of

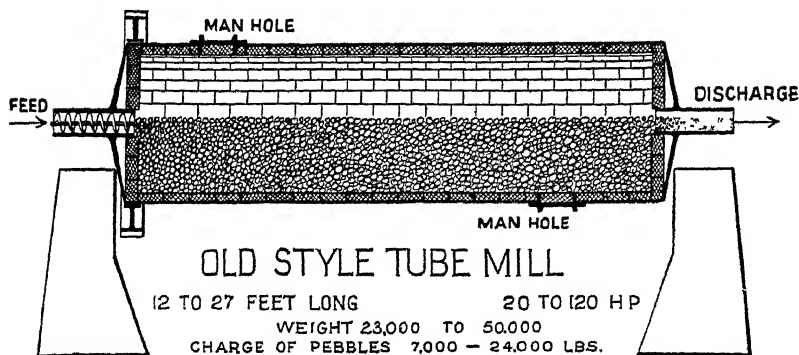


FIG. 1.

particles. The first mills were comparatively short, from 4 to 8 ft. long, then suddenly they were standardized at about 20 ft., probably by the builders of machinery, and the additional fact that they were more commonly used in the grinding of cement, which must be finished in one operation. The final conclusion at which I arrived was that the later practice of using a long tube-mill was wrong. My next construction of a tube-mill reduced it to 8 ft., and still later to 6 ft. At the feed end of the mill a certain energy was applied, which I will designate as a 1 lb.-pebble unit, which is necessary to crush a 0.25-in. unit of material. At the outlet end of the mill this same pound-pebble unit was still being employed even though the particle had been reduced to $\frac{1}{8}$, to $\frac{1}{16}$, to $\frac{1}{32}$ in. and so on in the same pro-

¹ *Trans.*, xxxix., 336 to 341 (1908).

portion of mass division, so that the 0.25-in. mass had been successively divided until the original particle was represented by more than 500 particles of $\frac{1}{32}$ in., upon which, or, at best, upon several of which, particles there was being used the same 1 lb. energy unit as was employed at the feed end of the mill. This would be equivalent to crushing 1-in. cubes of ore in a rock-crusher designed to reduce a mass containing several hundred 1-in. cubes. The latter would be ridiculed simply because the discrepancy is physically evident, but we go on with the still poorer practice of trying to reduce a $\frac{1}{510}$ particle with the same medium used to reduce the original unit mass, simply because we cannot see or feel the energy difference. It is equivalent to the return constantly of the 1-in. cube to the giant crusher in the hope that it may, as it eventually will, ultimately meet with some obstructions and be by chance further reduced. But during all this time energy is being expended in friction of moving parts without material division. On the other hand, it is hardly to be hoped that we can reach beyond mechanical means to obtain a required result, as would be the case if we endeavored accurately to adjust the cause to the effect desired, but we certainly can do better than to continue to employ the pound-pebble unit of energy when a 1-oz. unit will produce better results. This I have endeavored to do in the conical mill, in the different diameters of which are segregated forces proportional to the work to be performed; using the largest pebbles, the greatest peripheral speed, and the greatest gravimetric force upon the coarsest particles, gradually reducing them in the diminishing diameters of the cone, thus gradually diminishing all the forces commensurate, or proportionately commensurate, to the work required in reduction, though retaining the same number of grinding mediums. In other words, lines of force in the Hardinge mill converge at a point where the greatest power is needed and energy recedes in proportion to the force required in reduction, as illustrated in Fig. 2. Kick's law of power required for dividing particles of matter, although formulated by Kick 50 years ago, is still referred to in textbooks as defining the work performed in division and is as follows:

"The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies."

Apparently the principle in the inanimate machine is nearing that of the animate, and is a survival of the fittest, for it is an inherent mechanical property of the machine which controls the conditions whereby the largest bodies seek and maintain positions of greatest force and exert their power upon the weaker, if the latter are in the path of the larger; and it appears to be a case of the survival of the largest; *e.g.*, in the conical mill the largest pebbles or balls, the greatest superincumbent weights, and the greatest peripheral speeds all segregate towards the greatest diameter of the mill, likewise other sizes seek zones or positions in proportion to mass and weight acting in conjunction with gravity and central forces. The same rule holds

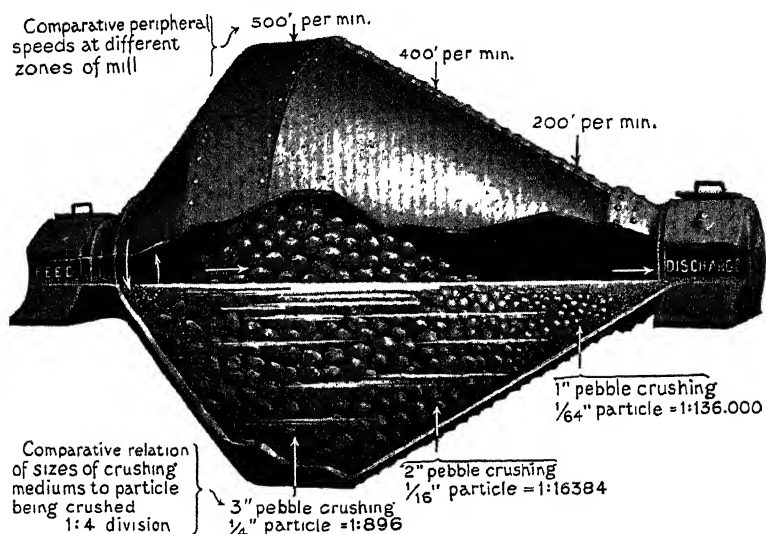


FIG. 2.

good in regard to the particles undergoing division, as illustrated in Fig. 2, in which it will be noted that the grinding mediums automatically adjust or classify themselves to the work to be performed, embodying a step- or stage-reduction within a single machine through a combination of percussion and attrition. Comparatively, a sledge-hammer is used upon a spike, a nail-hammer upon a nail, and a tack-hammer upon a tack, utilizing the mechanical forces more economically than would be the case in using a sledge-hammer without regard as to whether the blow was to be delivered upon the spike, nail or tack.

The illustration of the segregation of the large and small grinding bodies (Danish pebbles) shown in Fig. 3 is taken from a report of an independent engineer of international reputation in order to verify if possible my claims.

These general results are obtained in other systems of dividing or reducing by multiplying the number of machines; for instance, in rolls it is common practice to pass the successively

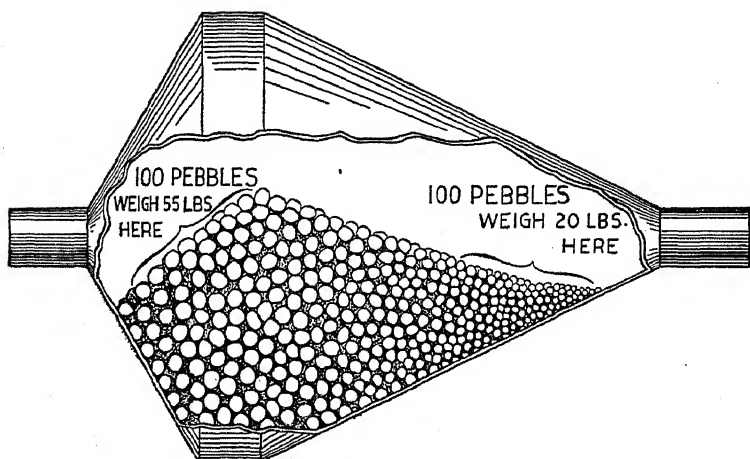


FIG. 3.

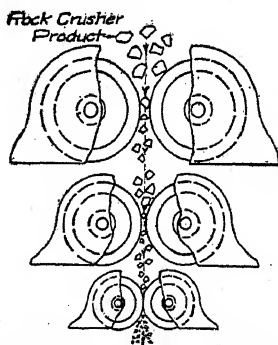


FIG. 4.

reduced material to rolls of different sizes, as illustrated in Fig. 4, although more often the poorer practice is resorted to of returning to the same roll the "oversizes" of its previous reduction, which must now depend for further division upon crushing (mashing) in a "choked feed"—an energy-wasting method.

The above remarks relative to the conical pebble-mill also apply to the conical ball-mill, which is similar in all respects

to the pebble-mill except that instead of using flint lining and pebbles, steel lining and balls are used to do the crushing. The latter is now coming into use in the place of stamps as a more economical device. The stamp to-day is gradually receding from its former prestige owing to its being neither an economical fine or coarse crusher, as the term is now understood. A few years ago 30-mesh was considered fine crushing, but it is so no longer. If one should insist upon its use as a fine crusher and to obtain mechanically commensurate results for power expended, the stamp should be reduced in proportional size to the work required and the screen-aperture should also be likewise reduced. It has taken us a long time to recognize the fact that to economize energy, the units of force must correspond to the units of matter being acted upon, as explained in Kick's law. In the breaking of a 2-in. cube of ore by a single drop of a 1,000-lb. stamp, we divide the mass into an approximate average size of 0.25-in. cubes, and perform reduction of 1 to 500. Assuming the energy has been well applied, the further reduction should be in the same ratio of energy to work, but instead of reducing the weight of the stamp to $\frac{1}{500}$ of the original, the work continues with the initial 1,000 lb. of energy—a sledge-hammer is now doing tack-hammer work.

The step-reduction mentioned above is further illustrated by the reducing sizes of stamps, shown in shadow, against the Hardinge mill, as in Fig. 5, in which the weight and number of the balls are proportional to the reduced size of stamps.

We have many times been asked to explain the action within the conical mill and a cause for its reasonably uniform product. The answer would resolve itself into the explanation of the automatic segregation of forces in proportion to mass being acted upon by different degrees of rotative energy. In addition to these natural automatic laws, the conical mill is subject to still further regulation by changing the axis to various inclinations from the horizontal, which will cause the finer particles to travel towards the outlet much more rapidly and consequently be subject to lesser action of the grinding bodies, as their travel will be assisted by gravity rather than displacement by the heavier bodies and the attendant crushing. A practical illustration of this action can easily be obtained by placing two glass funnels base to base after partly filling them

with gravel and fine dry sand, joining them with cement or adhesive tape, then evenly revolving with the axis horizontal; the experiment may also be tried having the axis at a slight inclination from the horizontal, as illustrated in Fig. 6. The result will be found to be curious and interesting mainly because unexpected.

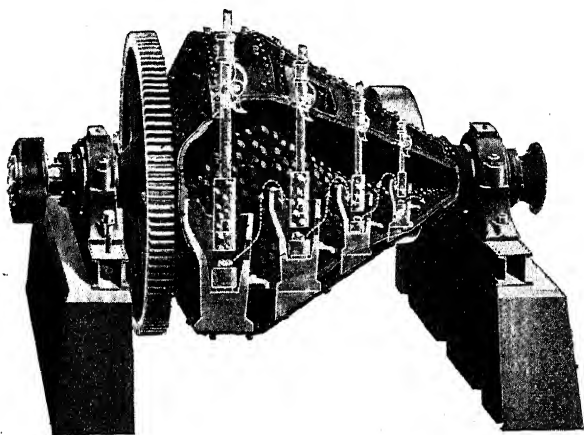


FIG. 5.

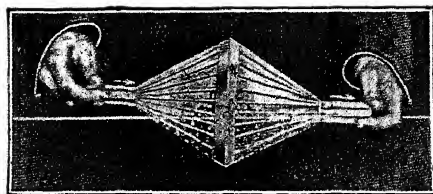


FIG. 6.

A feature of considerable importance in the Hardinge mill is the fact that the grinding bodies, whether of steel balls, flint pebbles, or large pieces of the ore itself, are utilized to a finality; thus there is no rejection before actual final consumption, no scrap-heap of costly and partly-consumed material, for after the first charging of the mill with its grinding mediums, the subsequent and desirable difference in sizes of grinding mediums is produced by the wearing away of the larger sizes. No grinding body is thus discarded because it is too small.

Lining.

The pebble-mills are lined with silex blocks throughout, or a combination of silex and ribbed steel or smooth plates, the plates being fitted with a special design of lifting-bar, which not only assists in lifting the mass of grinding bodies higher, affording greater impact in the central portion of the mill, but also prevents any slipping of the charge.

Speed of Mills.

Authorities vary widely as to the best speed for rotation of pebble-mills; in the case of the ordinary cylindrical mill (Fig. 1) the speed is rarely brought above 500 peripheral ft. per min., ranging between 400 and 500 ft. per min.; this speed, of course, is maintained throughout the total length (average) of 20 ft. In the case of the conical mill it can be more effectively operated at a peripheral speed of about 750 ft. per min. for the 8-ft. mill, but is maintained at this speed only for a length of about 20 in. instead of 20 ft. For the ordinary granular reduction, desired in concentration and other metallurgical processes, wherein a maximum of about 20-mesh and a minimum of slimes is the end desired, this speed of 750 peripheral ft. per min. (in the 8-ft. mill) is gradually reduced in proportion to the energy necessary for the further reduction of the previously divided particle. Thus in the one machine peripheral speeds—consequently the energies—vary from 200 to 750 ft. per min. in gradual stages or steps. It is a device wherein the same revolutions per minute produce a multiplicity of gradually changing peripheral speeds proportional to the diameter of the cone.

Competitive Test of Hardinge vs. Chile Mills.

The following figures in Table I. and Fig. 7 are the analysis of data furnished by a very large mining company which ran an 8-ft. Hardinge mill in direct competition with a 6-ft. Chilean mill.

The company was under the impression that the Chilean mill had the better of the argument, based upon gross tonnage fed, even though the power and water consumed (both costly items) were in favor of the Hardinge mill. Net tonnage was the economic feature, therefore also vastly in favor of the Hardinge mill.

TABLE I.—*Comparison of Results of a Hardinge Conical Mill and a Chilean Mill.*

Mesh	On 4mm.	On 8	On 12	On 20	On 30	On 40	On 60	On 80	On 120	On 200	Thru 200
Feed to both mills	38.5	51.3	7.6	1.9	0.4	0.3	
Product from Hardinge mill	0.1	1.7	12.5	13.0	15.2	11.1	14.4	7.5	21.5
Product from Chilean mill	..	4.1	16.2	11.1	7.6	3.9	3.3	4.4	9.7	5.4	30.3
Percentage total copper in feed...	24.29	63.40	9.02	2.02	0.51	0.76
Percentage total copper in Hardinge product	0.46		3.46	6.23	15.66	13.65	20.81	11.99	27.74
Percentage total copper in Chilean product.	..	0.98	5.45	5.26	3.49	2.00	2.98	3.25	12.31	10.31	53.97

In order to examine the results of this test properly, the cumulative percentage on each mesh was plotted for both machines, as shown on the lower half of Fig. 7. The line marked "Line of Ideal Product" is a uniformly graded product through 20- and all on 200-mesh, which will give a maximum extraction of copper from this ore, which was a disseminated sulphide.

As may be seen from the curves, the product from the Hardinge mill approaches this "ideal" line more closely than does that from the Chilean mill, and, is therefore, better suited for the economical extraction of its metal-content. The curves also show the Hardinge mill has finished 98.2 per cent. of its feed through 20-mesh, whereas the Chilean mill puts through this screen only 66.6 per cent. of the same feed. Of these quantities, the Hardinge mill has only put 21.89 per cent. through 200-mesh, as compared with 45.49 per cent. for the Chilean mill. In other words, the Hardinge mill gives 76.7 per cent. of its total feed as a product from which the maximum amount of copper can be extracted, while the Chilean mill gives only 36.3 per cent. between the same limits (through 20-mesh and on 200), as is shown in the upper half of diagram. Here the percentage on each mesh has been plotted, as shown, and from the figures on "Per Cent. of Total Copper" in Table I. the values at the right were calculated.

Considering the amount of copper contained in the two products, that from the Hardinge mill has 71.8 per cent. of its total copper-content in the material through 20- and on 200-mesh, as compared with 34.34 per cent. for the Chilean mill.

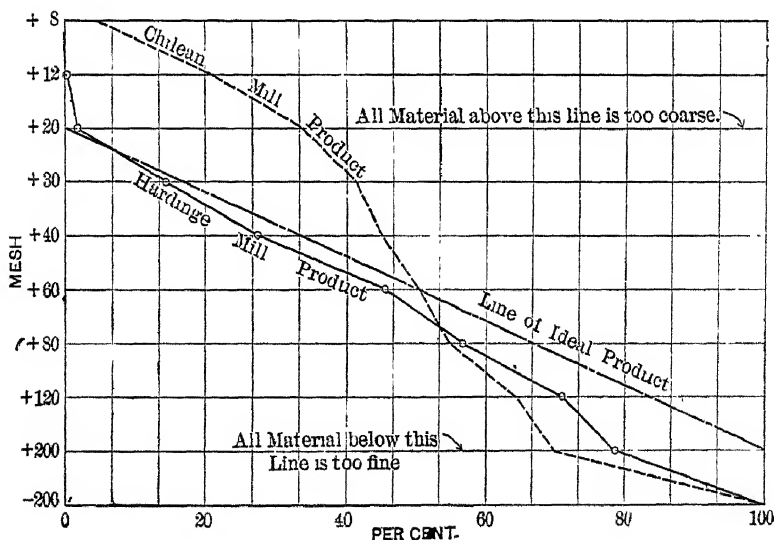
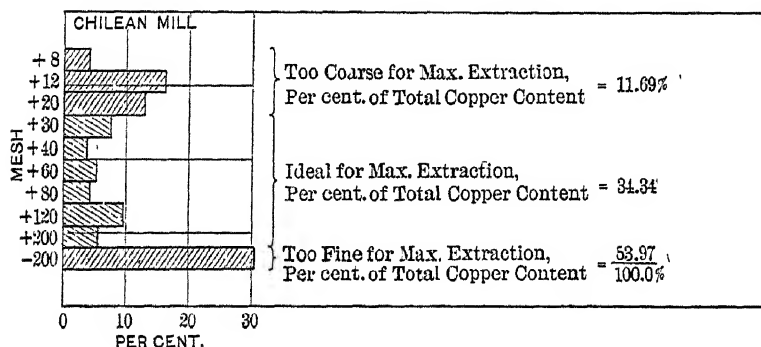
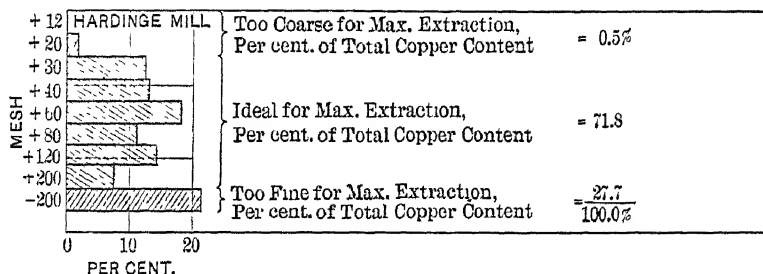


FIG. 7.

This result will materially affect the amount and cost of recovery.

As to the amount of water required, the Hardinge mill used only 19.25 gal. per minute per ton per hour put through 20-mesh, whereas the Chilean mill used 43.84 gal.² In a locality where water is scarce, this is a very important item. Although no mention is made of horse-power in the data given us, nevertheless, judging from other installations of the same character, the Hardinge mill requires less than 70 per cent. of that needed to operate the Chilean mill.

To recapitulate, the above data may be presented as follows :

TABLE II.—*Comparison of Products of Hardinge and Chilean Mills.*

	Hardinge Product.	Chilean Product.
Total tons through mill.....	71.5	83.5
Per cent. on 20-mesh (undesirable).....	1.8	33.4
Tons through 20-mesh (finished).....	70.21	55.29
Per cent. through 200-mesh (undesirable).....	21.5	30.3
Tons through 20- and on 200-mesh (desirable).....	54.84	29.99
Per cent. of copper in material on 20-mesh.....	0.46	11.69
Per cent. of copper in material through 200-mesh.....	27.74	53.97
Per cent. of copper in desirable material.....	71.8	34.34

These figures show that although the Chilean mill received a feed of 17 per cent. more than the Hardinge mill, the latter finished 27 per cent. more through 20-mesh, and has 82.5 per cent. more than the Chilean mill between the desirable limits of 20- and 200-mesh. The Hardinge mill also has over twice as much copper within these limits of economical extraction, using less than half the amount of water per ton; and presumably requiring two-thirds the horse-power.

Conical Ball-Mills vs. Cylindrical Ball-Mills.

A certain prejudice appears to exist against the use of ball-mills, particularly the older types which have mainly been successful on dry crushing, and to which Philip Argall³ refers in his article as follows:

“Ball mills are successful in dry crushing, and of little account in wet work, because of the heavy abrasion of balls, plates and particularly screens.”

² This amount of water for the Hardinge mill can be readily reduced to 4 gal. per min., or about equal weights of ore and water, or less, thus giving it a still greater advantage over the Chilean mill.

³ *Engineering and Mining Journal*, vol. xcv., No. 2, p. 109 (Jan. 11, 1913.)

Those who are not familiar with ball-mills might do well to consider the difference between the Hardinge and the German type probably referred to by Mr. Argall. These two distinct types of ball-mills should hardly be classed together any more

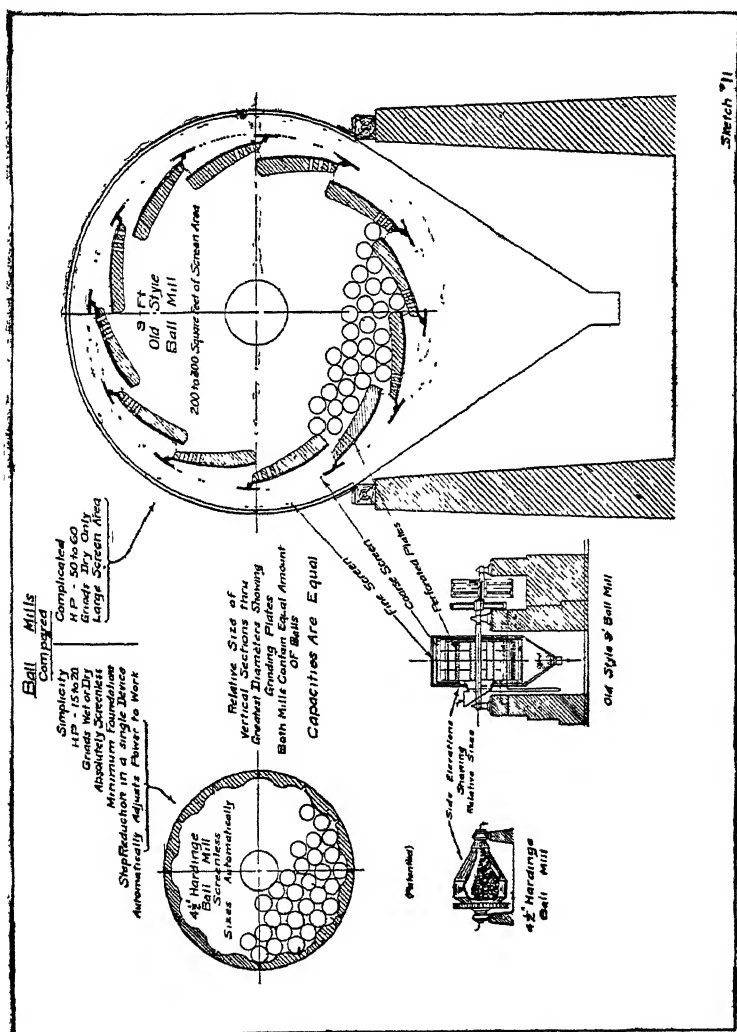


Fig. 8.

than a gyratory should be classed with the jaw type of rock crusher, simply because they both come under the general head of crushers. Fig. 8 graphically illustrates the difference in the two types of ball-mills above mentioned, which we further compare as follows :

TABLE III.—*Comparison of Hardinge and German Ball-Mills.*

	Hardinge Ball-Mill	German Ball-Mill
Size	4.5 ft.	9 ft.
Capacity	4 to 8 tons per hr.	4 to 8 tons per hr.
Power	15 to 17 h-p.	40 to 50 h-p.
Liners	Solid steel.	Perforated steel.
Screens	None.	200 sq. ft.
Operated	Wet or dry.	Dry only.
Ball charge.	4,000 lb.	4,000 lb.

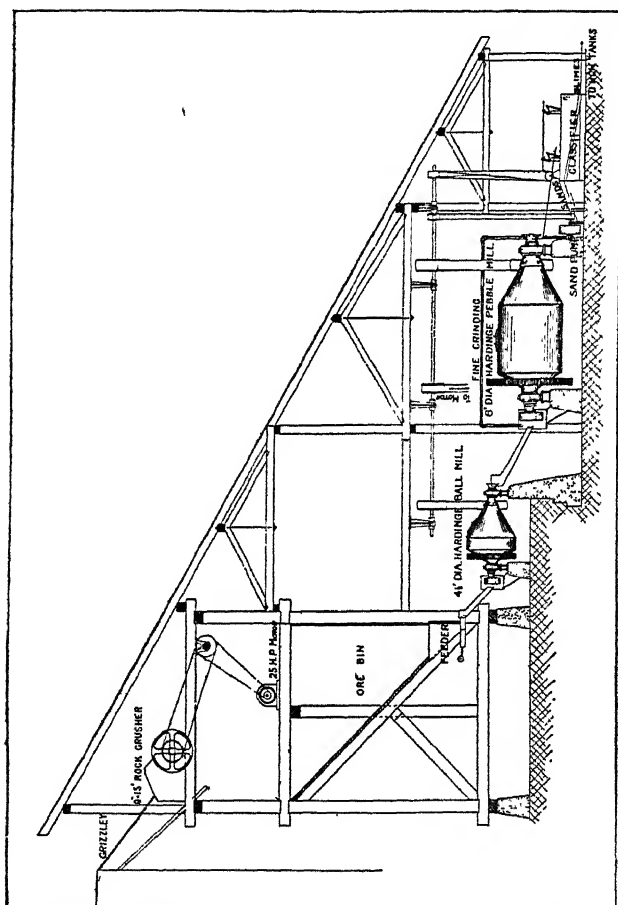
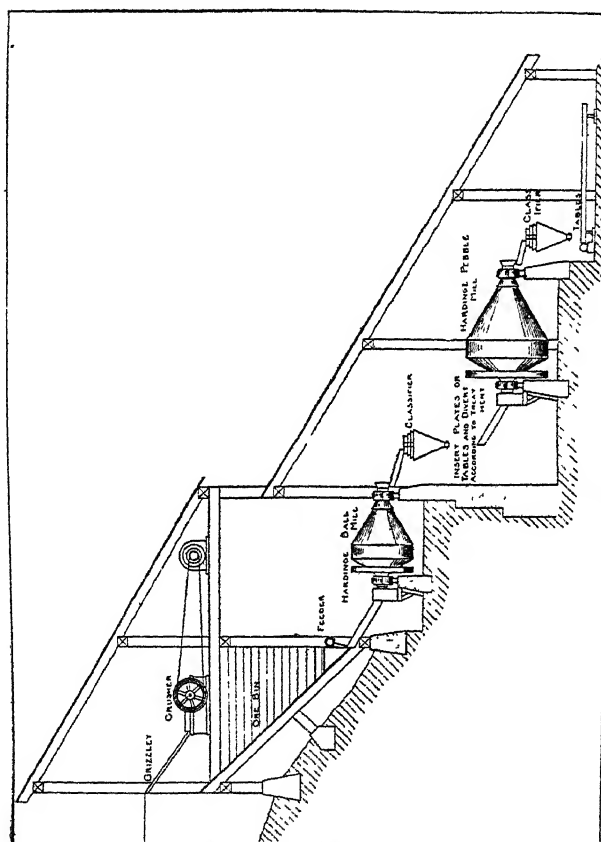


FIG. 9.

The German ball-mill has been in successful operation many years, particularly in Australia, though it has not found much use in America. The conical ball-mill as now constructed and used comes into direct competition with stamps owing to its extreme simplicity, and enters into a particularly simple flow-

sheet, shown in Fig. 9, producing a crushed material of the grades shown in Table IV., furnished by the Porcupine Gold Mining Co., of Porcupine, Canada. For the particular installation mentioned the consumption of balls and lining is given as less than 1 lb. per ton of ore ground. The material fed to the ball-mill, some of which exceeds 2 in. in diameter, is taken directly from a rock-crusher.



HARDINGE CONICAL MILL CO., 59 Church Street, New York
Fig. 10.

TABLE IV.—Results Obtained by Hardinge Conical Mill,
Porcupine, Canada.

Mesh.	On 10	On 20	On 40	On 80	On 100	On 200	Thru 200
Feed to 4.5-ft. Hardinge ball-mill. (Crusher product thru 1.5 in.)	88.56	4.17	7.0				
Product from 4.5-ft. Hardinge ball-mill	8.50	18.4	20.78	15.0	3.30	34.0	
Product from 6-ft. by 72-in. Hardinge pebble-mill				0.65	4.92	3.56	90.87

Metallurgical requirements calling for granulation with minimum of colloids or slimes are fulfilled along the lines flow-sheet shown in Fig. 10. We append results of this sty of installation furnished by the Beaver Consolidated Mine Ltd., of Cobalt, Canada, shown in Table V.

TABLE V.—*Results Obtained by Hardinge Conical Mill at Cobalt, Ont.*

Mesh	On 6	On 8	On 10	On 20	On 40	On 60	On 80	On 100	Th 10
Feed to 4.5-ft. Hardinge ball-mill, (Crusher product through 1.5-in. screen on 0.25 in.)									
Product from 4.5-ft. Hardinge ball- mill	28 6	30 4	13.7	17.7	4.9	1.8	1.1	0 1	1
Product from 6-ft. by 22-in. Hardinge pebble-mill				2.7	30.0	33 7	18 5	4 2	10

As further illustrative of the wide range of the conical mi Table VI. gives results of crushing in the same size (8 f of mill upon different classes of material, from flint-conglomerate ores of the Lake copper-district to the softer porphy copper-ores of Arizona. Naturally, vastly different tonnages are obtained according to hardness and other physical properties.

TABLE VI.—*Results Obtained by Hardinge Mill on a Variety of Ores.*

Mesh.	On 10	On 20	On 40	On 60	On 80	On 100	On 150	On 200	Thru 200	Ton- nage	H-
<i>Calumet & Hecla Mining Co.</i>											
Feed. . .	52.3	39.7	6.3	1.0	0.7
Product...	0.02	0.15	2.1	6.55	6.0	16.9	26.7	40.8	50	3
<i>Copper Range Consolidated Co.</i>											
Feed	58 0	26.2	14.8	0.9
Product	8.5	3.3	9.4	30.4	10 9	34 0	71
<i>Federal Mining & Smelting Co.</i>											
Feed	41.2	45.8	11.0	1.3	0.3	0.3	0.1	115	25
Product..	0.1	8 0	15.6	16.4	14.5	7.0	13.4	25.0
<i>Miami Copper Co.</i>											
Feed.....	25.3	32.7	10.1	1.8	0.8	0.7	0.8	0.4	7 3	180	3
Product..	2.2	13 8	15 5	8.9	7.7	8.5	5.3	38.1

Several types of conical mills are manufactured in different sizes for the following purposes:

(a) Granular grinding for concentration with a minimum of slimes, taking a product or feed passing $\frac{1}{4}$ - to $\frac{3}{8}$ -in. screen.

(b) Fine or slime crushing with a minimum of coarse, taking a feed of 0.25 in. or less.

(c) Ball-mills taking 2 in. and smaller cubes, replacing stamps, rolls and other coarse to fine crushers.

The horse-power required for these mills depends somewhat upon the charge of grinding bodies and the material undergoing disintegration, but is approximately:

	Horse-power.
4.5-ft. ball-mill.....	10-15
6-ft. by 22-in. granulating pebble-mill.....	15-18
6-ft. by 72-in. sliming pebble-mill.....	25-30
8-ft. by 22-in. granulating pebble-mill.....	30-35

The Hardinge mill has gone through the usual stages of competition and patent infringement which ordinarily follow the introduction of a successful device. The patents have been upheld by the U. S. Circuit Court and the U. S. Circuit Court of Appeals.

Notes on the Formation of Ferrites in Roasting Blende.

BY G. S. BROOKS, DEPUE, ILL.

(New York Meeting, February, 1913)

THE tendency of the oxides of such metals as aluminum, zinc, chromium, and calcium to form compounds at high temperatures with iron oxide is well established by past investigation. Data of this reaction, as developed in laboratory research, have been contributed by Burleigh, Hofman, and Wells, in America, and by many others abroad. A partial bibliography of the subject is appended. From these authorities we learn that the ferric oxide here has the function of an acid, giving in the case of ZnO a zinc ferrite, with a normal composition of ZnOFe_2O_3 .

This formation of zinc ferrite particularly affects the success of such processes as leaching or magnetic separation, and thus becomes practically important to the metallurgist.

But the meager existing data do not entirely agree; and quantitative results from muffle-practice are conspicuously absent. Furthermore, it being of some general importance to the zinc smelter in both roasting and reduction, the following few notes of operations on a commercial scale are contributed:

I. PHYSICAL PROPERTIES OF ZINC FERRITE.

As formed for the most part synthetically by investigators, zinc ferrite exhibits characteristic physical properties, such as octahedral crystals, dark color, and a density over 5, together with a slight magnetic attractability. In examining with the microscope many products of mechanical muffle-roasting kilns, it is, however, impossible to find individual grains which can be identified as consisting entirely of zinc ferrite. The variety of colors, especially where each grain of blende was mechanically free from marcasite, rather conveys the impression of a partial ferritization of the grain. One side or end, for example, may often have the hard, dense surface and dark red to black color of the ferrite, while the balance shows the usual light yellowish brown of a normal iron-free blende.

The hard, dense surface, resisting the entrance of lixivants, undoubtedly makes leaching more difficult. No tendency towards crystallization has ever been observed. This mottled coloration does not result from roasting the Western marmatites, the product from which always appears uniformly dark.

Commercially roasted ore contains ferrites of some little magnetic attractability.

II. CHEMICAL COMPOSITION.

For determining to what extent this chemical combination of the zinc and iron takes place, a suitable method consists in the use of a solvent which will remove the uncombined zinc oxide, but which does not attack the zinc oxide combined with the iron oxide. The amount of zinc thus dissolved being deducted from the total zinc, the remainder, contained in the residue, is chiefly zinc ferrite. But in commercially-roasted ore, a small amount of the zinc remains as a sulphide. To estimate this amount approximately, the total sulphur content of the roasted ore is first determined, after which the portion combined as SO_3 in sulphates is found and deducted, leaving as a balance the amount of sulphur present as sulphide.

This sulphide¹ I have usually found, in the case of muffle-furnace products, attached to both the zinc-bearing and the iron-bearing grains of the roasted ore, but in larger quantity with the zinc mineral, so that it is considered sufficiently accurate to compute the sulphide as ZnS .² However, in commercial roasting where the total sulphur of the ore drawn is in the neighborhood of 2 per cent., the sulphur present as sulphide rarely rises above 0.4 per cent.; so that any error in the assumption will be slight indeed. It is apparent that the amount of ferrites thus calculated would always be a little low, should any of the iron be combined with sulphur as sulphide.

It is well to note here that, in a finished kiln product, when the roasting has been thoroughly done, a large part of the sulphur present as sulphate is in calcium and magnesium sul-

¹ This applies to blende-marcasite mixtures and is based on the magnetic separation and subsequent analysis for sulphur of the roasted minerals.

² With ferruginous blende this may or may not be the case. See experiments of Jensch.

phates from the gangues.³ With the exception of possibly a few Western ores, then, the sulphur as zinc sulphate is relatively very low compared with the sulphur in the form of other sulphates.

As a solvent, Lowe's solution, containing 200 g. of NH_4Cl , 500 cc. of NH_4O_4 (sp. gr. 0.90), and 750 cc. of H_2O , was used. A 0.5-g. sample is digested in 50 cc. of this solution for an hour, being kept just below boiling, to prevent the driving off of NH_3 fumes. The residue is filtered and washed with the same solution. Any further treatment is found to bring only slight additional traces of zinc into the solution. The filtrate is then acidified and titrated with ferro-cyanide as usual.

A confirmation of the accuracy of this method was found in the roasting of an unusually pure Mexican blende. This ore showed beautiful crystallization and high transparency. Selected sphalerite from it gave upon analysis: Zn, 65.1; Fe, 0.10; and CaO, 0.2 per cent. After roasting, the whole of the zinc was soluble in Lowe's solution, as might be expected, since the unusually small proportion of isomorphous iron present could form only a negligible quantity of zinc ferrite. A later roasting of the same blende mixed with pyrite, from which all materials passing 40-mesh had been removed, showed but a trace of ferrites. Results are shown below:

Total S.	S as SO_3 .	Total Zn.	Soluble Zn.	Insoluble Zn.	Zn as S
Per Cent		Per Cent.	Per Cent.	Per Cent	Per Cent.
0.14	Trace.	78.2	77.9	0.3	0.2

The method appears, therefore, to meet the practical requirements of works-determinations.

III. FORMATION OF FERRITES.

In studying the conditions on the kiln hearths under which zinc ferrite results, there are two distinct ores to be considered; the one a mechanical mixture of iron- and zinc-bearing minerals, and the other a mineral in which the iron and zinc are in chemical combination. In the former case the contact of grains plays a

³ Fe_2O_3 does not react with CaSO_4 , forming ferrite, until $1,100^\circ \text{C}$. See Hofman and Mostowitsch, *Trans.*, xxxix., 646 (1908).

more important part than in the latter. Of course, for a given weight of material, the surface of possible contact increases inversely as the diameter to which the grains are crushed.

Especially with blende ores carrying large quantities of marcasite, the question arose as to whether the ferritization was not in some measure increased at the beginning by the cementation of a grain of partly roasted marcasite, fusing with its neighboring grain of blende, as they lay side by side in the kiln.⁴ The freezing curve of these two minerals, made by K. Friedrich, shows a eutectic composed of 5 per cent. of zinc sulphide and 95 of iron sulphide, and having a freezing point of approximately 1,150° C. Roasting temperatures are not often carried, in the early stages, as high as this; and, while the individual grain itself may at times, as for instance after rabbling, flash to this temperature, it is not often that incipient fusion sufficient to stick juxtaposed grains together occurs to any practical extent on the kiln hearths. To this may be added the evidence of the microscope, which shows but few grains rounded by fusion. An ore very high in marcasite will tend more to stickiness, and it is likely that practically all the iron-zinc matte thus formed will yield ferrites, much the same as marmatites. In these ores, the fines offer an opportunity for local overheating by flashing at the passage of a rabble. At such times they are burned with such rapidity that the result is often a shower of sparks, undoubtedly indicating a grain-temperature of more than 1,100° C. Kiln products sometimes show the larger zinc oxide grains bristling with these fine, fringe-like coatings; and, of course, unless a reasonable control of the upper hearth temperatures is maintained, the bed will become very sticky. In well-regulated kiln practice this does not often happen.

That calcium ferrite should furnish a possible binder does not appear likely, under kiln temperatures, and with the small quantities of lime present in well-jigged ores. The Western and Canadian ores, in which the blende often carries as much as 12 per cent. of iron in the crystals, present the most favorable conditions for the formation of ferrite.

1. *Progress of Formation.*—To learn at what stage this ferritizing action begins, and under what working conditions it might be controlled, requires something more than an analysis

⁴ Fulton, *Trans.*, xxxix., 584 (1908).

of the final products. Roastgraphs showing the progressive desulphurization and ferritization, under hearth conditions into which only a few variables were introduced at each roast, are given later with this end in view.

Inasmuch as the multi-hearth mechanical furnaces develop somewhat progressive reactions, by reason of the gradual advance of the ore through the varying physical and chemical conditions of the floors, simultaneously with the continuous change of the ore itself in composition due to the advance of desulphurization, it is obviously easier to chart the action for a furnace in which the ore-charge itself does not move. The resulting products of such work, done on a small muffle hand kiln, checked closely with those of the larger mechanical furnaces; and this progress of the reactions as noted is instructive.

It is generally agreed that zinc sulphides in roasting form intermediate products, such as normal and basic sulphates. The amount and make-up of these depend, in a measure, upon the existing temperatures, pressures, and gas compositions of the hearths. However, it is very probable that the reaction for the formation of zinc ferrite in practice does not occur through the contact of iron sulphate and zinc oxide, as largely formed by laboratory experiments. The reason is, that the iron mineral has little tendency to form the sulphate at the working temperatures, which exceed the decomposition point of the latter by at least 200° C. It has been suggested that the reaction might be made possible by the conversion of either the iron or the zinc for a moment to a somewhat nascent state as their desulphurization proceeds. But the fact that the oxides themselves combine on heating renders this suggestion less plausible.

It appears most probable that the tendency of these oxides to combine as ferrite begins with the zinc, either as zinc sulphate (the iron being oxide), or with both metals as oxides, or with the desulphurization of any zinc-iron matter present. I have found that the two roasted oxides, when mixed in a very fine condition, do form ferrites to an appreciable extent, notwithstanding some of the earlier observers to the contrary. The statement of W. Hommel,⁵ "It is noteworthy that after the largest portion of the sulphur has been removed at the lowest possible temper-

⁵ *Metallurgie*, vol. ix., No. 9, pp. 281 to 296 (May 8, 1912.)

ature, the ore could be exposed to the highest heat obtainable in a roasting furnace without any ferrite being formed," has not been borne out, either by our practice or by laboratory observations. In fact, the two oxides, from which the last of the sulphur has been roasted, continue to increase in ferrites for several hours. Ferrites have been formed in quantity on separately roasted oxides, afterwards re-mixed, where there is no longer danger of local flash overheating, at a bed temperature of not over 950° , a much lower figure than given by Hommel.

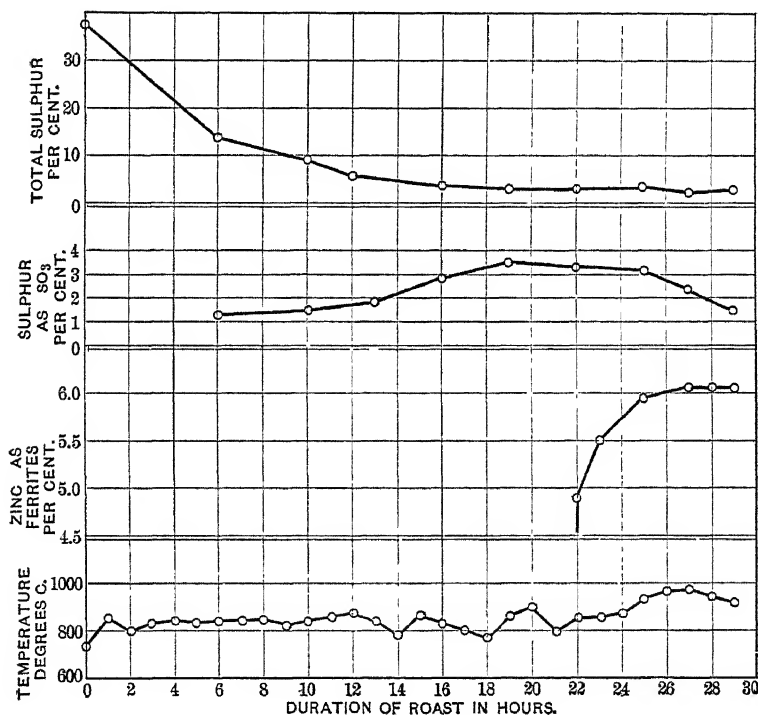


FIG. 1.—ROASTING TEST ON "K" JIG CONCENTRATES.

Zinc sulphate and ferric oxide, when roasted together in a laboratory assay muffle at a low heat, were found to yield a high percentage of zinc ferrite, which is in accord with the results of Mostowitsch, although the temperature on commercial sizes was near 900° C.

It seems doubtful, on observing the variations in SO_3S (see curves in the diagrams, Figs. 1 and 2) during the hand kiln roasts, that very much of the $ZnSO_4$ was dissociated by the reaction with

the Fe_2O_3 . The reason for this belief is, that the rate of dissociation, theoretically, should be much increased as the ferrite is being formed. Yet we find, on the contrary, that the sulphates actually increase as the ferrites are increasing, and the rate of formation of the latter decreases (as shown by the flattening of the curves) as the sulphates are broken up. It seems most probable that the increase in sulphate-sulphur results from the formation of zinc sulphate, and not from the sulphates contributed by the gangues, since these are sulphatized much

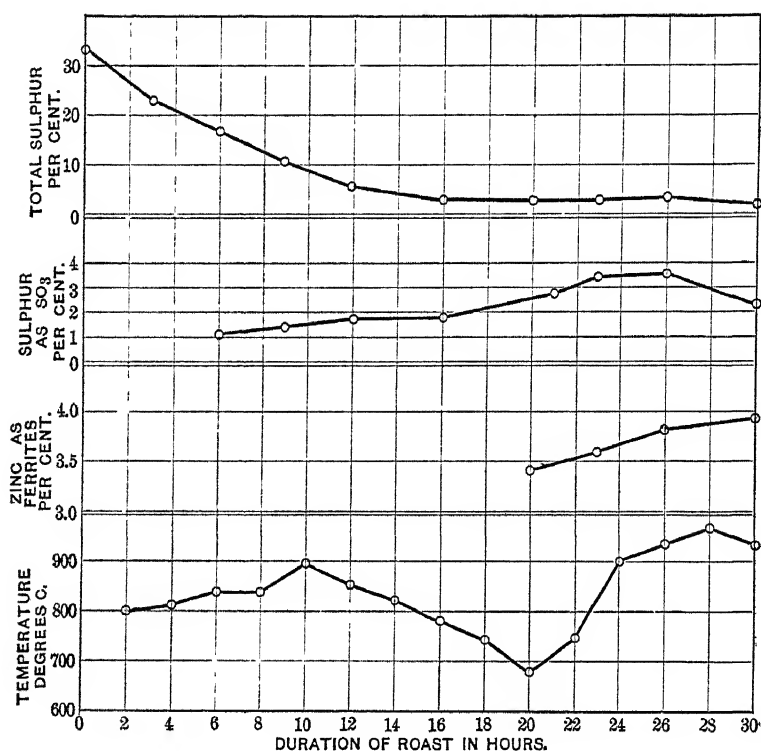


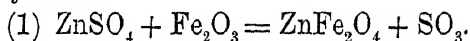
FIG. 2.—ROASTING TEST ON "V H" JIG CONCENTRATES.

earlier in the roast. The degree to which this reaction takes place will in the end, of course, depend upon the difference in temperature between this formation of ferrite and the decomposition of zinc sulphate. Should the former be as high as the latter, by reason of poor contact, etc., it is not likely that any considerable reaction can be expected.

Composition of Ferrites.

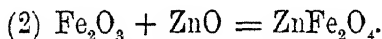
It is certain that the kiln products contain zinc ferrite of greatly varying composition, ranging in works products all the way from a normal ferrite at one end of the series to an extremely basic one at the other, so that its formula cannot be at all times accurately written. Furthermore, no definite relation between the iron-zinc ratio of the green ore to this composition has been discovered.

2. *Reactions.*—In the kiln, then, as this zinc sulphate breaks up from the normal through the basic salt to the oxide, at the same time in physical contact with a neighboring grain of iron oxide, we may have:

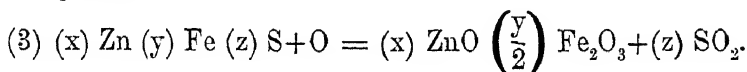


(At hearth temperatures, SO_3 is usually dissociated into SO_2 and O .)

At times, this reaction has been shown to be reversible, the zinc rather than the iron being sulphatized, because, in such a case, the temperature is above the decomposition point of iron sulphate and below that of zinc. (The ferrites are actually broken up.)



Or, resulting from the zinc-iron matte formed at the beginning of the roast on the top hearths by local overheating and fusion, we may have:

*Tests.*

For the curves, Figs. 1 and 2, which are typical of a series of 15 tests made on the above-mentioned hand kiln, the temperatures were taken at 30-min. intervals with a thermo-couple in the ore and samples were drawn hourly with a long-handled scoop from the ore bed. The effects of area of contact, temperature, and time are taken up separately below. The ores used in each of the two charts were from different mines, but both were blende-marcasite jig products. Their composition was as follows:

	Mine "K"	Mine "V H."
Zn.....	35.1	47.7
Fe.....	19.6	10.7
CaO.....	1.51	1.63
Pb.....	0.55	0.35
S.....	37.74	34.7

The behavior of the sulphates is of interest. Note their gradual increase preceding their breaking up at the close of the roast.

3. *Surface Factor*.—Before discussing the variation in amount of ferrites formed as a result of a more or less intimate mechanical contact of grains, it should be noted that with the perfect molecular contact of the isomorphous iron of the Western or British Columbia blendes, the size of grain has little bearing. Under this condition, other variables, such as length of roast, temperature, etc., affect the action.

The ordinary ores, as prepared at zinc-smelting plants, having been dried and crushed to pass a 2-mm.-opening screen, show the greatest variation in their amount of dust. In many cases, this fine material is produced before the ore reaches the plant. The screening and crushing which follow its arrival may add from 7 to 10 per cent. by weight to the "through 60-mesh" fines. It is clear that these are contributed by the drying, crushing, screening, and intermediate handling. A sizing test made on the ores from mine "C" is given below. The limiting screen for this material was a little larger than was later used.

Mesh	Grams	Cumulative Weight	Cumulative Per Cent
+ 4.....	222	222	11.1
— 4 + 6.....	347	569	28.5
— 6 + 8.....	229	798	39.9
— 8 + 20.....	557	1,355	67.8
— 20 + 60.....	507	1,862	93.1
— 60.....	138	2,000	6.9

During the roasting, considerable decrepitation of the grains naturally results, and the two following analyses clearly illustrate the action in the kilns on the carefully sampled and screen-analyzed lot from mine "K":

Mesh	Weight.	Cumulative Weight.	Cumulative Per Cent.
+ 4.....	13	13	0.7
— 4 + 6.....	329	342	17.2
— 6 + 8.....	367	709	35.6
— 8 + 20.....	777	1,486	74.6
— 20 + 60.....	415	1,901	95.4
— 60.....	92	1,993	4.6

The product of this roast was :

+ 4.....	3	3	0.2
— 4 + 6.....	75	78	5.2
— 6 + 8.....	105	183	12.2
— 8 + 20.....	427	610	40.8
— 20 + 60.....	645	1,255	83.9
— 60.....	240	1,495	16.1

The contact between grains of equal diameters is, of course, small in comparison with the mixed sizes. Roughly speaking, however, for equal weights, materials which will pass through 60-mesh screens present relatively much greater areas of contact than do the coarser pieces, so that it is at once reasonable to suppose that the finer materials under identical conditions of roasting tend to form the iron-zinc ferrite more easily. This is shown in the table given below :

“K” Concentrates.

Mesh.	Total Sulphur.	S as Sulphates.	Total Zn.	Zn as Oxide and Sulphate.	Zn as Sulphide.	Zn as Ferrite.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
+ 10.....	3.64	2.82	45.6	21.9	1.6	2.1
— 10 + 40.....	1.84	1.44	45.0	39.6	0.8	4.6
— 40 + 80.....	2.88	2.62	48.6	41.2	0.4	7.0
— 80.....	3.90	3.70	41.4	30.4	0.4	10.6

“C” Concentrates (fines).

Mesh.	C. Weight	Total Zn.	Soluble Zn	Total S.	S as SO ₂ .	Zn as ZnS.	Zn as Ferrite.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
+ 10..	1.9	27.2	14.5	6.9	3.0	7.8	4.9
— 10 + 20..	15.6	36.6	26.8	3.4	2.7	1.4	8.4
— 20 + 40..	49.8	45.1	36.4	3.1	2.9	0.4	8.7
— 40 + 60..	71.6	44.7	35.0	3.6	3.3	0.6	9.1
— 60..	28.3	35.2	19.5	4.0	3.6	0.8	14.9

“L G W” Concentrates.

+ 10..	36.8	Sample	lost
— 10 + 40..	63.2	63.4	58.8	2.1	1.6	1.0	3.6
— 40 + 60..	78.6	64.2	56.8	2.0	1.7	0.6	7.6
— 60 + 80..	88.3	62.6	54.4	2.1	1.8	0.6	7.6
— 80..	16.7	55.6	45.9	2.4	2.2	0.4	9.3

These show a marked increase with the finer sizes. The "K" concentrates had been crushed to a limiting screen of 2.5 mm. The "C" concentrates had been crushed through a 2-mm. gravity screen, with choke feed-rolls cleaning up the work of the screen. The "L G W" concentrates represent about average sizes for Middle West sulphides.

4. *Rabbling of Ore.*—In addition to the area of contact, the frequency with which the ore is moved grain upon grain within its bed has a decided bearing upon the extent of ferritization. Mechanically rabbled kilns have frequently intermittent raking periods, which may be spaced as far as two hours apart. With these, the opportunity of a grain of iron and one of zinc mineral combining may be more than in the continuous raking type. The advantage, however, of the latter over the former in lessening the ferrites is somewhat offset by the additional comminution of the grains through more frequent stirring, and possibly also by the fresh surfaces being turned up for combination with neighboring grains. This last has never been indicated sufficiently to receive much weight.

IV. TEMPERATURE EFFECT IN FERRITIZATION.

That the rise in temperature increases the tendency for combination of the iron and zinc is natural. It has also been noticed that the higher temperatures sometimes give ferrites towards the more basic end of the series. Below is shown the effect of increased temperatures on ore roasted simultaneously in two hearths of a hand kiln.

"K" Concentrates.

Hearth.	Hours.	Average Temperature.	Zn as Ferrite.
Top.....	30	840° C.	6.3
Bottom.....	30	930° C.	8.2

V. LABORATORY DATA.

The table given below was compiled from supplementary laboratory data. The question of increase in ferritization with rise in temperature, extension in time of roasting, and fineness of grains is clearly illustrated. Roasts Nos. 2 and 3 were carried

out in the carbon-resistance furnace, in which the temperature was under excellent control, while Nos. 1, 4, 5, and 6 were made in the muffle of an ordinary fire-assay furnace.

"C" Jig Concentrates (Fe, 21 per cent.).

Test, No	I.	II	III.	IV.	V.	VI.
Size	Through 50.	Through 50.	Through 50	On 40.	On 60.	On 60.
Time of treatment....	(1) 6 hr.	2 hr.	6 hr.	6 hr.	6 hr.	6 hr.
Average temperature.	600-850	900° C.	1,000° C.	Same	as	No. 1
Furnace.....	(2) Muffle	C. R. Furn.	C. R. Furn.	Muffle	(2)	
Rabbling	30 min.	None	None	Every 30 min.		
Total Zn.....	47.7	50.3	50.0	49.9	57.3	55.9
Soluble Zn.....	38.4	38.8	37.6	45.3	49.1	46.2
Comb. Zn.....	9.3	(3)	(3)			
		11.5	12.4	4.6	8.2	9.7
Total S.....	2.3	Trace	Trace	Trace	Trace	Trace
S as SO ₃	2.3	Trace	Trace	Trace	Trace	Trace
Zn as sulphide.....	0.0	0.0	0.0	No shines	
Material treated.....	Green ore	Prod. I	Prod. I	Green ore	

NOTE :—(1) Starting at 600°, slowly rising to 850° C.

(2) Ore in roasting dishes.

(3) Note basic composition of this ferrite.

Column I. of the test shows the results on fines of green ore roasted with a gradually increasing heat.

Column II. shows the results of extending the roasting of the product of I. in an electric furnace at an elevated temperature, for 2 hr. more.

Column III. shows a continuation of this work.

The test in column IV. shows the tendency of sized materials to decrease in ferritization because of limited contact-areas.

Columns V. and VI. show the increase, even with sized materials, as the grains decrease in diameter.

Further data on the effect of the time element show as follows :

Kind.	Time.	Total S.	S as SO ₃ .	Total Zn.	Sol. ZnO and Sulphate.	ZnS.	Ferrite Zn.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	P. C.	P. C.
"C" Concentrates..	26 hr.	5.58	4.84	44.0	39.6	1.5	2.9
"C" Concentrates..	56 hr.	2.80	1.70	42.2	36.0	2.2	4.0
"VH" Concentrates	60 hr.	1.84	1.70	57.0	51.3	0.3	5.4
"VH" Concentrates	30 hr.	3.52	3.22	54.7	50.3	0.5	3.8

Roasting Marmatites.—To illustrate the quantitative effect of molecular contact of the iron and zinc in a typical Western marmatite, an analysis of the product of a 7-hearth mechanical kiln is herewith given :

“E Z” Concentrates (Colorado).

Total Zn.	Soluble Zn.	Total S.	S as SO ₃	Zn as ZnS.	Comb Zn.	Fe.
Per Cent 49.1	Per Cent 35.2	Per Cent. 2.6	Per Cent 1.6	Per Cent. 2.0	Per Cent. 11.9	Per Cent. 14.0

There is practically no free pyrite in the original green ore charged, and 25 per cent. of the total zinc is combined.

VI. RÉSUMÉ.

The foregoing observations seem to indicate :

(1) The possibility of some control over ferritization in works roasting by reducing the contact surfaces through proper preliminary treatment.

(2) That highly oxidizing atmospheres in the kiln hearths may permit quicker roasting and lower temperatures, both of which lessen the tendency to combine. This, however, must always allow for the exigencies of acid manufacture with its strong SO₂ gas requirements.

(3) That frequent rabbling may aid, when not excessive, to the point of grinding ore grains into fines.

(4) That the combination as a ferrite on the hearths may occur with the two minerals either as oxides, or as a combined matte, or from the reaction of ZnSO₄ and Fe₂O₃.

(5) That true marmatites yield a high proportion of ferrite under all conditions of commercial roasting.

(6) That the formation temperature in practice is somewhat below 1,000° C.

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The Role of Certain Metallic Minerals in Precipitating Silver and Gold.*

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(New York Meeting, February, 1913.)

INTRODUCTION.

WHILE the reducing action of organic matter, of ferrous sulphate, and of hydrogen sulphide has frequently been invoked to account for the deposition of native gold and silver from ore-forming solutions, the high efficiency in this respect of certain of the metallic minerals which form the ore itself has not been sufficiently recognized. If a little finely powdered chalcocite is placed in a test-tube and 2 or 3 cc. of a dilute solution of gold chloride are poured over it, two or three shakes of the tube suffice to remove all the gold from the solution. The color of the solution changes from yellow to pale green, showing that copper is dissolved simultaneously with the precipitation of metallic gold.

If a piece of chalcocite is hung in a dilute aqueous solution of silver sulphate ($\frac{1}{10}$ normal), a coating of silver begins immediately to form on the mineral, and within a few moments a beautiful silver tree has developed, similar to that formed when metallic zinc is immersed in a similar solution.

When similar experiments are conducted with other sulphides and with certain arsenides and sulph-arsenides, especially those known to be commonly associated with native silver in ore deposits, it is found that these minerals differ greatly in their efficiency as precipitants, especially in their effect on silver solutions. The tests made with most of the minerals were qualitative only, but the reactions of a silver sulphate solution on chalcocite and niccolite were submitted to quantitative study to determine the nature of the reactions involved. Quantitative studies of the reactions with other minerals are now in progress

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in the laboratory of the U. S. Geological Survey; but as the phenomena seem to have an important application in the science of ore deposits, particularly in secondary enrichment, it has been thought best to outline the field of investigation in this preliminary paper and to make the main results immediately available to economic geologists.

The earliest important work on the precipitation of gold and silver by metallic minerals was that of W. Skey,¹ printed in 1871 in a New Zealand publication. His work appears until recently to have escaped the attention of economic geologists; it blocked out, however, one of the most interesting and promising fields of research in the science of ore deposits.

Another line of investigation inaugurated by Skey² involves the measurement of the electrical potentials developed when two metallic minerals are connected as poles of a galvanic circuit and immersed in some conducting liquid which does not react actively with them. This line of research has recently been followed by Gottschalk and Buehler³ and R. C. Wells.⁴

These writers have arranged the commoner metallic minerals in the order of the electrical potentials developed when they are in contact with a conducting liquid, and Wells concludes that "such potentials simply measure in an electrical way the oxidizing power of the solutions, the mineral playing a very insignificant part. . . . The potentials in metallic salts solutions depend entirely upon the nature and concentrations of the solutions."

It is now recognized that most reactions of oxidation and reduction may be effected by the use of the electric current in place of chemical agents, and, conversely, that an electric current may readily be produced by a proper arrangement of the

¹ Skey, W., On the Reduction of Certain Metals from Their Solutions by Metallic Sulphides, and the Relation of This to the Occurrence of Such Metals in a Native State. *Transactions and Proceedings of the New Zealand Institute*, vol. iii., pp. 225 to 231 (1871).

² Skey, W., On the Capability of Certain Sulphides to form the Negative Pole of a Galvanic Circuit or Battery; also, On the Electro-motive Power of Metallic Sulphides. *Transactions and Proceedings of the New Zealand Institute*, vol. iii., pp. 222 to 225, 232 to 236 (1871).

³ Gottschalk, V. H., and Buehler, H. A., Oxidation of Sulphides. *Economic Geology*, vol. vii., No. 1, pp. 15 to 34 (Jan., 1912).

⁴ Wells, R. C., *Journal of the Washington Academy of Sciences*, vol. ii., pp. 514 to 516 (1912). In abstract.

components of any one of these reactions. As a consequence, by measuring the relative strengths of the currents developed when various sulphides are successively connected as parts of a galvanic circuit, it is possible to obtain a measure of the relative rate at which oxidation or reduction takes place. Such methods of study may prove useful in determining the relative efficiency of various minerals in reactions of the type described in this paper, but careful analytical studies are essential to their apprehension and for the present appear to form a more profitable line of research.

EXPERIMENTS WITH SILVER SULPHATE SOLUTION.

In these experiments the minerals were selected so as to be as free as possible from admixtures of other minerals. Considerable variation is known to exist in the composition of natural minerals referred to the same species, and such variations may influence greatly their efficiency as precipitants of the precious metals. As a matter of record, therefore, data are given in a foot-note,⁵ showing the source of the material used. The experiments were conducted at ordinary temperatures. Pieces about 0.25 cc. in size were taken and were immersed in small evaporating dishes in 20 cc. of $\frac{1}{40}$ normal solution of silver sulphate. The solution was neutral at the beginning of the tests. The progress of the reaction was observed at appropriate intervals.

Silver sulphate was chosen as the reagent in these experiments because it is well known that the solutions of the oxidized zone in sulphide ore deposits are sulphate solutions and also because the sulphate is much more soluble than either the

⁵ Alabandite, La Ilucha, Puebla, Mexico, U. S. N. M. No. 19565; Arsenopyrite, State of Washington. Gives profuse arsenic fumes before blowpipe; Bornite, Aspen, Colo., U. S. N. M. No. 81851; Chalcocite, Butte, Mont.; Chalcopyrite, Loon Creek district, Idaho; Cobaltite, Buffalo mine, Cobalt, Ont., coll. of F. L. Hess, U. S. G. S.; Covellite, Butte, Mont.; Enargite, Butte, Mont., U. S. N. M. No. 85106; Galena, Joplin, Mo.; Jamesonite, Cornwall, England, U. S. N. M. No. 12500; Marcasite, Joplin, Mo.; Millerite, Gap mine, Lancaster, Pa., U. S. N. M. No. 83959; Niccolite, Cobalt Lake mine, Cobalt, Ont., coll. of F. L. Hess, U. S. G. S.; Orpiment, Mercur mine, Mercur, Utah, U. S. N. M. No. 84708; Pyrite, Gilpin County, Colo.; Pyrrhotite, from pegmatite, Woodstock, Md.; Realgar, Mineral Creek, near Tacoma, Wash., U. S. N. M. No. 84752; Smaltite, Cobalt district, Ont.; Sphalerite, Joplin, Mo.; Stibnite, Arkansas, coll. of F. L. Hess, U. S. G. S.

chloride or the carbonate, the relative solubility in water at 25° C. being approximately as follows: ⁶

Silver chloride.....	0.002
Silver carbonate.....	0.033
Silver sulphate.....	8.01

The experiments reveal notable differences in the efficiency of the sulphides, arsenides, and sulph-arsenides treated as agents for the reduction of metallic silver from dilute aqueous solutions of its sulphate, and various reaction products in addition to metallic silver are developed in most cases.

In tentatively grouping the minerals tested according to their efficiency it is recognized that the speed of the reaction could only be roughly estimated by the methods adopted and that some of the minerals carried impurities which may have had an important influence on their activity.

Very strong: Chalcocite and niccolite.

Strong: Covellite, enargite, bornite, tennantite, alabandite, and possibly cobaltite.⁷

Moderate: Smaltite, marcasite, pyrrhotite, chalcopyrite, and arsenopyrite.

Weak or inactive: Cinnabar, stibnite, pyrite, galena, millerite, sphalerite, jamesonite, orpiment, and realgar.

EXPERIMENTS WITH OTHER SILVER SOLUTIONS.

The silver sulphate solution used in the experiments just described was neutral. In order to determine the effect of acidity upon the reactions, a solution of silver sulphate was rendered acid by the addition of sodium acid sulphate (NaHSO_4) and its action on chalcocite, chalcopyrite, and niccolite was observed. No marked difference was noted in the speed of action of the acid and neutral solutions. A minor feature of interest was the development of delicate feathery silver crystals on the chalcocite immersed in the acid solution, while that immersed in the neutral solution formed brilliant spangles.

In order to approach more closely the conditions actually found in certain ore deposits a solution was made up which

⁶ Cox, A. J., Abegg, R., *Zeitschrift für physikalische Chemie*, vol. xlii., p. 11 (1903).

⁷ The "cobaltite" used was found to contain considerable amounts of nickel and its activity may be due partly or wholly to this metal.

except for its silver content corresponded in composition to the vadose water collected by Weed⁸ from the second level of the Mountain View mine, at Butte. This water was dripping from the roof of the drift and was depositing chalcantite. The artificial solution had the following composition:

$\text{CuSO}_4 \cdot 5 \text{H}_2\text{O} = 180 \text{ g. per liter.}$

$\text{ZnSO}_4 \cdot 7 \text{H}_2\text{O} = 2 \text{ g. per liter.}$

$\text{Ag}_2\text{SO}_4 = \text{less than } 4 \text{ g. per liter.}$

Chalcocite, bornite, covellite, galena, chalcopyrite, and niccolite were treated with this solution and silver appeared to be precipitated as rapidly as from the neutral silver sulphate solution. With the neutral silver sulphate solution galena precipitated only very minute amounts of silver after many days' immersion. With the cupriferous silver solution the action was more rapid, though still slow as compared with the other minerals tested. In general, the presence of copper in the solution appeared to facilitate rather than to retard the silver precipitation.

A simple experiment with solutions of silver benzol sulphate was performed to demonstrate the rôle of hydrolytic action in reactions of the kind under investigation. This organic salt of silver is very soluble in water; if chalcocite is immersed in a saturated solution of the salt, silver is precipitated only with extreme slowness; if, however, the solution is diluted with about 30 volumes of water silver begins to deposit rapidly and the reaction continues until silver has been completely removed from solution. In the equation given on page 231, representing the reaction of silver sulphate solution on chalcocite, no account is taken of the part which the elements of water play in the changes involved. That water does enter into the reactions is evidenced by this experiment.

QUANTITATIVE EXPERIMENTS.

Niccolite and Silver Sulphate Solution.

The so-called niccolite used in these experiments has the composition:

⁸ Weed, W. H., *Geology and Ore Deposits of the Butte District, Mont. Professional Paper No. 74, U. S. Geological Survey*, p. 101, analysis F (1912).

	Per Cent.
Ni	28.26
Co.....	12.25
Fe.....	0.46
As.....	53.02
S	5.89
	<hr/> 99.88

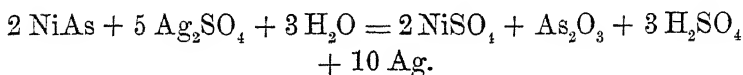
Weighed quantities of the finely powdered mineral were digested in dilute silver sulphate solutions containing known amounts of silver sulphate. Action began promptly. Free silver was deposited and the solution became green. After the lapse of several hours the gray residual mineral together with the silver deposit was collected on a filter. After deducting the weight of the silver the mineral residue was found to represent 35.12 per cent. of the niccolite treated. It consisted of cobalt, iron, arsenic, and sulphur, distributed as follows:

	Per Cent.
Co.....	12.25
Fe.....	0.46
As.....	16.52
S	5.89
	<hr/> 35.12

The composition of this unattacked residue obtained from the so-called niccolite corresponds to the formula for cobaltite, namely $\text{Co}(\text{As}(\text{S}))_2$.

The weight of silver deposited with the residual cobaltite amounted to 260.1 per cent. of the total weight of the mineral tested.

The portion of the mineral dissolved by the silver sulphate solution consisted only of nickel and arsenic. The nickel amounted to 28.26 per cent. and the arsenic to 36.50 per cent. of the original mineral. Since these results correspond to one atom of arsenic for every atom of nickel dissolved, it is apparent that the material attacked by the silver sulphate solution is true niccolite (NiAs). The arsenic was present as arsenious acid. Since the dissolved nickel, dissolved arsenic, and deposited silver are proportionally, nickel, one atom; arsenic, one atom, and silver, five atoms, the reaction between true niccolite (NiAs) and dilute silver sulphate solution may be expressed by the equation



From these considerations the so-called niccolite may be regarded as consisting of 64.76 per cent. of true niccolite and 35.12 per cent. of true cobaltite. Microscopic study of a polished piece confirmed the conclusion that it was a mixture of two minerals. That silver sulphate solution may be used advantageously to throw light on the nature of certain complex minerals is apparent, for by its agency two distinct mineral species, cobaltite and niccolite, have been quantitatively parted. In the original specimen, which contains both cobalt and nickel, apparently as inseparable companions, it is noteworthy that the silver sulphate solution assigns all the cobalt to the sulpharsenide, cobaltite, and places all the nickel in the simple arsenide, niccolite. The pyritous structure of true cobaltite may account for its failure to yield to the oxidizing action of dilute silver sulphate solution.

Chalcocite and Silver Sulphate Solution.

That free silver is deposited if metallic sulphides are digested in a silver nitrate solution has been mentioned by several observers, but their statements concerning the products formed lack unanimity. For instance, Heumann⁹ observes that from artificial cuprous sulphide (Cu_2S) free silver is deposited along with a gray powder, which he considers to be silver sulphide. Schneider¹⁰ corroborates Heumann. S. Meunier¹¹ states that he observed free sulphur among the products of the action of silver and gold solutions on several metallic sulphides.

In view of these divergent statements concerning the action of sulphides on silver solutions, it seemed advisable to determine the action of chalcocite on dilute silver sulphate solution. In no case was free sulphur formed, but the sulphur of the chalcocite was retained and held in combination as silver sulphide.

Total Ag deposited = 255.14 per cent. of chalcocite decomposed.

S in residue = 18.34 per cent. of chalcocite decomposed.

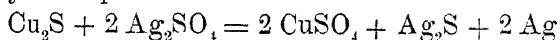
Cu dissolved = 75.81 per cent. of chalcocite decomposed.

⁹ *Berichte der deutschen chemischen Gesellschaft*, vol. vii., p. 1680 (1874).

¹⁰ *Poggendorff's Annalen der Physik und Chemie*, vol. clii., p. 471 (1874).

¹¹ *Comptes rendus de l'Académie des Sciences*, vol. lxxxiv., p. 638 (1877).

These figures correspond to: sulphur, one atom; copper, two atoms; and silver, four atoms, and the reaction may be expressed by the equation



While water does not appear in this equation it is clear from the experiment already cited on page 228 that it plays an essential part in the reaction. In a dilute silver acetate solution chalcocite acts in like manner.

Covellite containing 66.06 per cent. of copper and 33.87 per cent. of sulphur, immersed for four days in a dilute silver sulphate solution, yielded only 54.97 per cent. of its copper. This loss of 83.2 per cent. of copper and the deposition of an equivalent amount of silver as free silver and silver sulphide suggest that a more intimate structural relation exists between chalcocite and covellite than is indicated by the conventional formulas, Cu_2S and CuS , which are used respectively to represent the molecular masses of these two minerals. The study of the constitution of covellite is now in progress.

Experiments with Gold Chloride.

The minerals used were from the same localities as those used in the tests with silver. The experiments were conducted at ordinary temperatures. Pieces about 0.25 cc. in size were taken, and were immersed in small evaporating dishes in 20 cc. of gold chloride solution carrying about 7 g. of gold per liter. The solution was slightly acid to start with. The progress of the reactions was observed at appropriate intervals.

The gold precipitated by the action of certain of the minerals was mainly dark brown and porous, and the reaction continued until all the gold of the solution had become exhausted. The gold developed on other minerals, on the contrary, was yellow and compact, and when the piece was wholly plated this coating seemed to protect the mineral from further action. For this reason it is not possible to classify the minerals according to their activity in precipitating gold as closely as was possible in the case of silver. Under natural conditions the sulphides would seldom become completely coated with gold and the inhibitory action due to such complete plating would not be felt.

All of the metallic minerals tested that were efficient as precipitants of silver were also efficient as precipitants of gold.

Galena, pyrite, stibnite, and millerite, which were inefficient as precipitants of silver, were active as precipitants of gold.

Sphalerite and cinnabar acted only weakly in precipitating gold.

Effects of Thin Coatings of Cuprous Sulphides.

The occurrence of very thin films of gray secondary chalcocite on chalcopyrite is a common phenomenon in the upper portions of many ore bodies. A specimen from Gilpin county, Colo., showing such films was treated with silver sulphate solutions. The chalcopyrite precipitated silver only with extreme slowness but the thin chalcocite coating was a very efficient precipitant. The reaction forms, in fact, a convenient means of identifying such gray coatings as probably chalcocite when they are too thin to be detached and tested.

Peacock-colored coatings are also common on the chalcopyrite of some veins and such coatings may be readily produced artificially by placing a piece of chalcopyrite for a few moments in contact with an iron nail in a solution of copper sulphate. A piece so tarnished will precipitate silver and gold with great readiness, whereas the uncoated chalcopyrite acts only very slowly. It is clear, therefore, that thin films of cuprous sulphides on chalcopyrite or other minerals may be almost as efficient as larger bodies of such sulphides in precipitating the precious metals.

APPLICATION IN ORE DEPOSITS.

Association of Native Silver with Copper Ores.

The results recorded above find their principal counterpart in nature in the phenomena of secondary enrichment in ore deposits. The subject is a large one, and it is only necessary here to cite a few examples to show the influence of such phenomena upon the distribution of values in ore deposits.

If chalcocite exercises such a strong precipitative action upon the precious metals when these are carried in neutral or acid solutions, we should expect that the precious metals would be re-precipitated as soon as they reach the upper part of the chalcocite zone. Although most of the great copper deposits of the world in which chalcocitization has been observed carry only low values in gold and silver in the original ore, such evi-

dence as is available indicates that the chalcocite has exerted a precipitative influence upon the precious metals.

In the copper veins of Butte, Mont., there are present nearly all of the copper minerals which these experiments have shown to exert a strong precipitative action on silver and gold. Here the conditions should have been almost ideal for testing the efficiency of such action under natural conditions. Unfortunately, the precious metal content of the original ores is very low and little detailed information is available in regard to the occurrence of precious metals in the upper portions of the copper veins. In the Butte folio appears the following statement: "Native silver has been found in both the copper and the silver mines. It is the only silver mineral that it has been possible to recognize in the copper veins," a statement that suggests the efficiency of the copper minerals in reducing silver salts in solution to the metallic state.

According to Weed,¹² "Native silver is common in the silver veins and also in the oxidized portions of the copper veins. It has been observed in mossy aggregates and coatings, in fracture planes in glance [chalcocite] ores, and in bornite. Wire silver and fibrous silver also occur in cavities in bornite and glance [chalcocite], particularly in the upper levels of the Parrot mine."

Another line of evidence consists in a comparison of the average silver values in the upper and lower portions of certain veins. Weed states¹³ that for the upper levels of the Gagnon mine, at Butte, the ore carried an average of 3 oz. of silver to each 1 per cent. of copper, whereas the average silver content of the ores extracted from lower levels in 1905-06 was only half as great: namely, 1.5 oz. of silver to each per cent. of copper.

It appears probable, therefore, that the precipitative action of chalcocite and other sulphides of copper played a part in the localization of the silver values in the upper portions of the Butte copper veins.

The intimate association of bornite, chalcocite, and native silver has been noted by the writer in the Up-to-Date mine near

¹² Weed, W. H., *Geology and Ore Deposits of the Butte District, Montana. Professional Paper No. 74, U. S. Geological Survey*, p. 80 (1912).

¹³ *Ibid.*, p. 63.

Caribou, Boulder county, Colo. The rich ores of this mine carry chalcocite, bornite, native silver, and calcite (non-manganiferous) as their principal minerals, with subordinate amounts of covellite. The ores of this type thus far obtained were found within from 50 to 100 ft. of the surface and occur as small veinlets in and as replacements of altered pyroxenite. Some of the veins are an inch or so in width, and one of them yielded a specimen of native silver 6 by 8 in. by 0.25 to 1 in. thick. From this size they grade downward to exceedingly minute veinlets less than 1 mm. wide. The irregular form of the smaller veinlets shows that they probably developed by metasomatic replacement of the rock proceeding outward from a very minute fracture. The irregular mode of association of the native silver and the sulphides indicates that they were contemporaneous crystallizations. Covellite when it occurs is intergrown with bornite or chalcocite, bears no relation to vugs or cracks and evidently crystallized with the other sulphides.

There is little doubt that the rich silver-copper veins of this mine are the result of downward sulphide enrichment, the materials being supplied by primary veins carrying galena, sphalerite, chalcopyrite, and calcite, and other veins carrying chalcopyrite, pyrite, and quartz which are also exposed in these workings. The detailed evidence of such origin is not germane to the present discussion. In most of the natural occurrences previously described the native silver was deposited distinctly later than the sulphides, simulating closely the conditions of the experiments which are here reported. In the present case the copper sulphides and the native silver were precipitated together. The chemistry of the process as regards silver precipitation is believed to be essentially the same in either case.

Occurrences similar to that in the Up-to-Date mine are not uncommon. Beck¹⁴ described a similar association of minerals in the copper shales of Germany. He says:¹⁵ "In addition to the small grains [of sulphides] there also occur in the Kupferschiefer fine bands, mostly parallel to the bedding, of bornite and chalcocite, and along bedding-planes and cross-fractures, coatings of chalcocite, bornite, chalcopyrite, and native silver."

¹⁴ Beck, *Lehre von den Erzlagerstätten*, 3d ed., vol. ii., p. 155 (1909).

¹⁵ Translation by E. S. B.

These instances, while by no means exhausting the literature, suffice to show that the association of silver with chalcocite, bornite, and related copper minerals has been frequently noted. Doubtless the presence of native silver in intergrowth with such copper minerals will be found to be a common phenomenon as more copper ores are studied microscopically in the polished sections.

Association of Native Silver with Cobalt and Nickel Arsenides.

The experiments showed that niccolite, and, to a lesser degree, so-called cobaltite and smaltite, were comparable with chalcopyrite, bornite, and covellite in the strength of their reducing action upon an aqueous solution of silver sulphate.

The frequent association of native silver with cobalt and nickel arsenides is too well known to require lengthy comment. It is perhaps sufficient to cite as an example the Cobalt district of Ontario. Dr. S. F. Emmons,¹⁶ in describing this district, says:

"The more common minerals of the rich vein deposits are the arsenides of cobalt and nickel, smaltite with some chloanthite, cobaltite, and niccolite, associated with native silver. Less frequent are native bismuth, the silver minerals pyrrargyrite, proustite, dyscrasite, and argentite, the nickel sulphide, millerite, with occasional mispickel and tetrahedrite. The ordinary sulphides pyrite, galena, and zinblendite are occasionally found in the wall rocks, but apparently do not form an essential part of the deposits. The gangue minerals are calcite, with a little quartz, but both are in relatively subordinate amount in the rich parts of the veins."

"It is generally recognized that the native silver is of distinctly later deposition than the cobalt-nickel arsenides—indeed, the evidence in the mines is most conclusive. . . . Very often minute cracks may be seen crossing the cobalt mineral and gangue rock which are filled with films of native silver, and the flake or sheet form in which the silver is so often found shows that it has grown in such cracks."

In the experiments conducted with niccolite and so-called cobaltite and smaltite, one or more reaction products, in addition to native silver, were abundantly developed. Although the nature of these other products has not been determined, it is suspected that some of them are silver salts, and it is noteworthy, therefore, that several other silver minerals besides native silver are present in the cobalt veins.

¹⁶ Emmons, S. F., Cobalt District of Ontario. In *Types of Ore Deposits*, pp. 141 and 143 (published by *Mining and Scientific Press*, 1911).

Associations of Native Gold with Sulphides.

Without touching the question of the apparent preference manifested by gold for certain sulphides in deposits that are clearly primary, it is possible to cite numerous instances where the sulphides seem to have exerted a reducing influence during the process of secondary enrichment.

Reno Sales¹⁷ has found distinct and fairly good-sized particles of native gold on the surface of chalcocite crystals from the 1,000-ft. level of the Leonard mine, at Butte.

In describing secondary enrichment in the pyritic deposits of Huelva, Spain, Finlayson says:¹⁸

“The lower limit of the gossan is sharp and well-defined, and the line of contact between gossan and sulphide ore is sometimes marked by an earthy zone carrying considerable values in gold and silver. This has been described by J. H. L. Vogt, who has pointed out that it represents a concentration, during secondary enrichment, of the traces of gold and silver in the original ore, the precious metals dissolved in ferric sulphate being precipitated by the reducing influence of the pyrite.”

SUMMARY AND CONCLUSIONS.

The preliminary experiments which are here described show that certain sulphides, arsenides, and sulph-arsenides of copper, nickel, and cobalt precipitate metallic silver very efficiently from dilute aqueous solutions of silver sulphate. As the waters descending through the upper portions of most sulphide ore bodies are known to be sulphate waters, similar precipitative actions would be expected under certain natural conditions. The frequent association of silver in ore deposits with chalcocite and bornite, and particularly with niccolite and cobaltite, minerals which in these experiments were among the most efficient precipitants, warrants the belief that such reactions are of importance in secondary enrichment of ore bodies.

The more common sulphides, such as pyrite, galena, and sphalerite, were relatively inactive as precipitants of silver from aqueous solutions of its sulphate.

The quantitative results obtained with niccolite and chalcocite indicate that the essential chemical changes in reactions of

¹⁷ Cited by Weed, *Professional Paper No. 74, U. S. Geological Survey*, p. 79 (1912).

¹⁸ Finlayson, A. Moncrieff, *The Pyritic Deposits of Huelva, Spain. Economic Geology*, vol. v., No. 5, pp. 409 to 410 (July-Aug., 1910).

this type are due to oxidation through the hydrolytic action of water. It is apparent, therefore, that certain water solutions may act as potent oxidizing agents below the ground-water level.

Unlike gold, silver forms a large number of natural compounds. It is probable that the presence of certain other substances in the vein solutions, notably arsenic and antimony, may so modify the reactions that silver is precipitated not as native metal but as a compound such as polybasite or proustite. The conditions under which such compounds are formed forms an allied and very important field of research.

The experiments indicated that nearly all of the sulphides and arsenides common in ore deposits were capable of reducing gold from a solution of its chloride, although important differences in the rapidity of the precipitation were observed with different minerals. Most of the minerals that are especially efficient as precipitants of silver are also effective precipitants of gold, and a number of other minerals such as galena, pyrite, stibnite, and millerite that are inefficient in precipitating silver are efficient in depositing gold.

It is known that the waters descending through the upper portions of sulphide ore bodies universally carry chlorides, and it is probable that these chlorides have effected the solution of the gold. It is probable, therefore, that phenomena similar to those exhibited in the experiments with gold chloride solution play an important part in secondary enrichment in gold.

While the phenomena here described find their most immediate application in secondary enrichment, it is perfectly possible that such reducing effects of the sulphides may be responsible in part for the primary association of the precious metals with certain sulphides in preference to others. It is recognized, of course, that certain mineral associations in ore deposits are probably the result of processes analogous to differentiation in rock magmas, but it is quite possible that other associations, such as the apparent preference of gold for chalcopyrite and tetrahedrite rather than for pyrite in deposits carrying these three minerals, may be due to differences in the reducing power of these sulphides themselves. Light could probably be thrown upon this point by the investigation of the reducing effect of various sulphides upon silver and gold salts dissolved in solu-

tions having the composition of certain deep mine waters. The effect of increase of temperature on the reactions should also be studied.

It is a generally recognized fact that the purity of alluvial gold is greater than that of the veins in the neighborhood. This superiority in fineness has generally been explained by the well-known fact that silver is more readily soluble in natural waters than gold, and is by them removed from the natural alloy, thus increasing its purity. Mr. Lindgren has recently discussed this matter at some length in a report on the Tertiary gravels of California,¹⁹ and has presented a large number of statistical data leading to the same conclusion. It has been thought by certain geologists that this refinement of the gold was accomplished by solutions circulating through the gravels themselves, but Mr. Lindgren states that "so far as the Tertiary gravels of California are concerned, the conclusion of the writer is that solution and precipitation of gold have played an absolutely insignificant part." Under the conditions of the experiments here reported it was found that nearly all of the metallic minerals common in precious metal deposits were capable of precipitating gold, while a much smaller number, and these not the most common ones, were active precipitants of silver. When it is remembered that the source of the placer gold is the oxidized zone of the original deposit, and that the gold may have been dissolved and redeposited several times within the vein before erosion carried it into the alluvium, it seems probable that such selective precipitation may be a factor in this natural refining of gold.

¹⁹ Lindgren, W., The Tertiary Gravels of the Sierra Nevada of California. *Professional Paper No. 73, U. S. Geological Survey*, pp. 68 to 70 (1911).

The London Mine, Mosquito Mining-District,
Park County, Colo.

BY CHARLES J. MOORE, DENVER, COLO.

(New York Meeting, February, 1913.)

It seems a duty incumbent on the older members of the Institute to set forth in permanent form some of the results of their experience for the benefit of the younger members; this is the principal object of the present paper, which is the result of a recent examination requiring somewhat unusual geologic investigation.

History.

The London vein was first discovered by float and occasional outcrops in the year 1873, four years before the great lead carbonate discoveries of Leadville, but the mine was not definitely located and opened until two years later, when the present locations, Mother, Paris, London, and Hard to Beat, were surveyed for patent.

Since 1875 the London mine has been worked almost continuously until 1911, when the force was reduced to a few men only. In its early life the ore was assumed to be free-milling gold-quartz, although lead was a constituent of the vein throughout, and much useless expenditure was made in the erection and operation of a free-gold stamp-mill, with plates and tables for saving concentrates. The vein then opened was in the porphyry-limestone contact, a siliceous lead-ore, clearly most amenable to reduction by lead-smelter.

The losses due to this process led in time to the close of the mill. For 20 years past, the mine has been worked under lease and the ore shipped to the lead-smelters in Denver or Salida.

In January, 1912, I examined the mine to ascertain whether it was advisable to run a cross-cut tunnel some 4,200 ft. in length to intersect the main vein at a depth of 650 ft. below the present lowest workings. Finding a greatly-faulted zone, a close determination of the geological conditions became necessary to the solution of the problem.

Topography and Geology.

The London mine is located upon London mountain, which is a short spur of the Mosquito range extending in an easterly course 2 miles from the crest of the range, and to a height of 13,160 ft. above sea-level. The hill proper is oval in form, isolated, and a very prominent feature in any view of the range from the eastern side; it rises to 2,000 ft. above Mosquito gulch on the east, 1,200 ft. above the gulch on the south, 1,000 ft. above the glacial amphitheaters on the west and north, with steep slopes from 30° to 40° , covered for the most part with rock-slide on the narrow summit ridge and west slopes.

The regional geology is described by the late S. F. Emmons¹ in his report on the Leadville Mining-District, of which this is the eastern part, the London mine being 7 miles ENE. of the city of Leadville on the eastern or Platte River drainage-slope of the Mosquito range. The main portion of the district is on the western slope or Arkansas River watershed.

The results of the most recent geologic investigation of Alma district, completed during the last summer, are now in process of publication as a bulletin by the Colorado State Geological Survey.

The general course of the Mosquito range is north and south, and the orographic forces creating it resulted in several great faults approximately parallel to the range in course and all with eastern uplift, culminating in the Mosquito fault, passing 1.75 miles due west of the London mine, with an upthrow of 7,500 ft. This movement followed by erosion has brought to the present surface on the crest of the range, Middle Carboniferous grits with intrusive dikes and sheets of monzonite and other porphyries, identical with those of the Leadville mines.

On the summit of the range these beds dip to the eastward towards the mine at 15° , and are underlain by the entire series of Lower Carboniferous, Silurian, and Cambrian limestones, shales, and quartzites forming the bold escarpments on the western side of the range, surrounding the glacial amphitheaters at the head of Iowa and Big Evans gulches.

At a point S. $71^{\circ}51'$ W., 3,920 ft. from the mouth of the South London tunnel, a vertical bore was made by diamond-drill under direction of C. F. Carnahan and W. C. Mosher, who

¹ *Monograph XII., U. S. Geological Survey (1886).*

kindly showed me the log and cores. Excluding the various sheets of eruptives, these cores showed the total thicknesses of Lower Carboniferous blue limestone, 200 ft.; parting quartzite (calcareous and siliceous shales and quartzite; transition-beds between Lower Carboniferous and Silurian), 25 ft.; Silurian white limestone, 170 ft. In these two limestones are the principal ore-deposits of the Leadville district, and the blue limestone of Leadville is identical with that developed in the London mine as hereafter described.

At 2.5 miles NW. from the London mine another great fault starts on the western slope of the range at the Mosquito fault, and courses SE. through the western slope of London mountain and beyond for a total distance of 13.5 miles, ending in an anticlinal fold. This fault is known as the London fault, taking its name from the mine, where it has an anticlinal eastern upthrow of 2,000 ft., thus bringing the blue limestone of the Lower Carboniferous into contact with the Archæan granite and schists, Fig. 1.

For many years past the London vein has been reported to lie in the actual crevice of the fault-fissure, although contrary to the statement of Professor Emmons, who says "the ore-deposits follow the stratification planes" of the upper part of the Silurian limestone. Engineers have examined the mine from time to time, but no one has published the results of such examinations, and none of the private reports to the owners submitted to me contained a correct account of the actual ore-deposits.

In view of the fact that the nearest fault-fissures to the west contain true veins, it was advisable to test the London fault at various points to ascertain whether this also was mineralized. This was done under my direction at four points on the South London tunnel-level, as shown on the general plan of workings, Fig. 2, which also shows surface-contours referred to sea-level as datum, and the boundaries of mining-claims comprising part of the property and surroundings. Bores Nos. 1, 2, 3, and 5 located the line of the fault on the 12,000-ft. plane for about 2,000 ft. in length. Its location on the surface could not be determined with absolute accuracy on account of the slide-rock and few outcrops, but it is at no point probably more than 30 ft. from the location shown on Fig. 2. It thus appears that the

fault pitches vertically at the northern end of the London claim, and thence southward the pitch is easterly, reaching a minimum inclination from the horizontal of 62° in the center of that claim. Section C on Plate VIII.² erroneously shows this fault with a western pitch of 75° , but this pitch agrees approximately with the West London fault as I have located it.

The bore-holes developed the entire thickness of the blue limestone and parting quartzite, uptilted against the London fault so that the strata vary from a vertical dip in the oldest or northern part of the mine to 77° westerly in the southern part. At 900 ft. west of the main London fault, the grits overlying the blue limestone formation are exposed in the South London tunnel with a westerly dip of 39° , thus indicating approach to

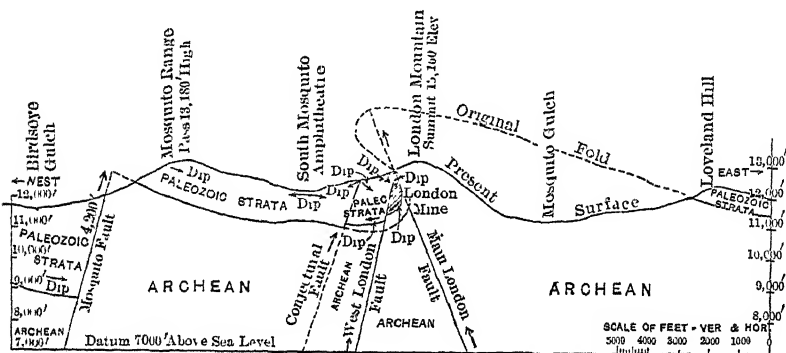


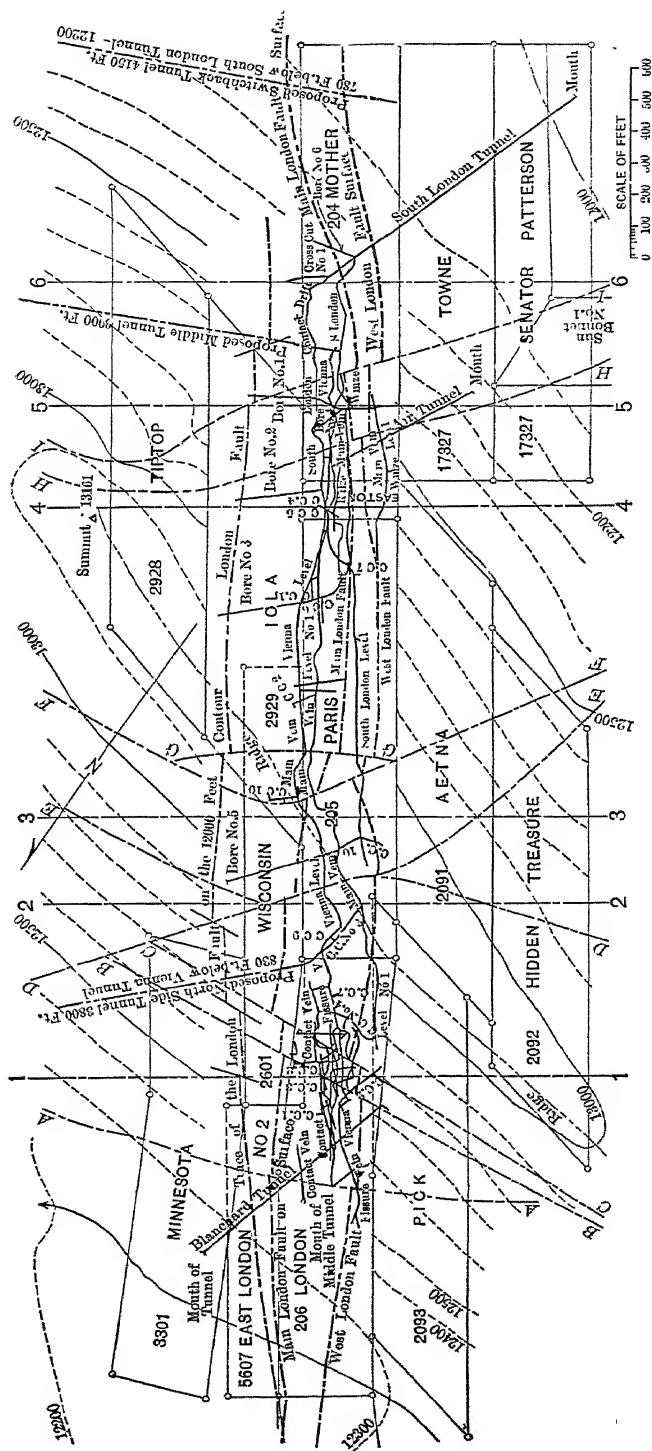
FIG. 1.—GEOLOGICAL SECTION THROUGH LONDON MOUNTAIN, SHOWING FAULTING.

the syncline and hypothetical fault shown in the geological section, Fig. 1.

The workings disclose also another fault, approximately parallel to, and a short distance west from, the main London fault. This is designated on Figs. 1, 2, and 4 to 9 as the "West London," and, though starting at the surface either vertical or with a steep western inclination, changes into a westerly pitch as depth is gained, and thus corresponds closely with that plotted by Professor Emmons, as above stated.

The fault marked "conjectural" on Fig. 1 is required by a short anticlinal exposure on the western ridge of London mountain and the eastern dips of outcrops in the South Mosquito amphitheater.

² *Monograph XII., U. S. Geological Survey (1886).*



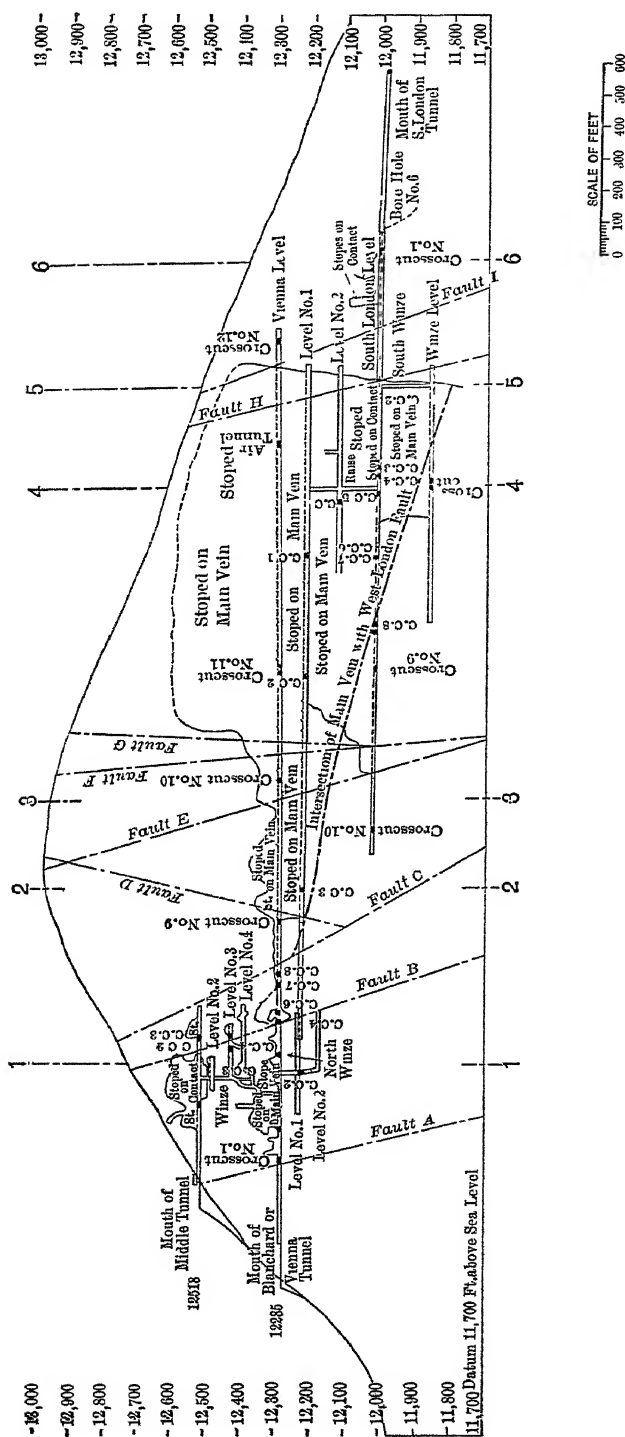
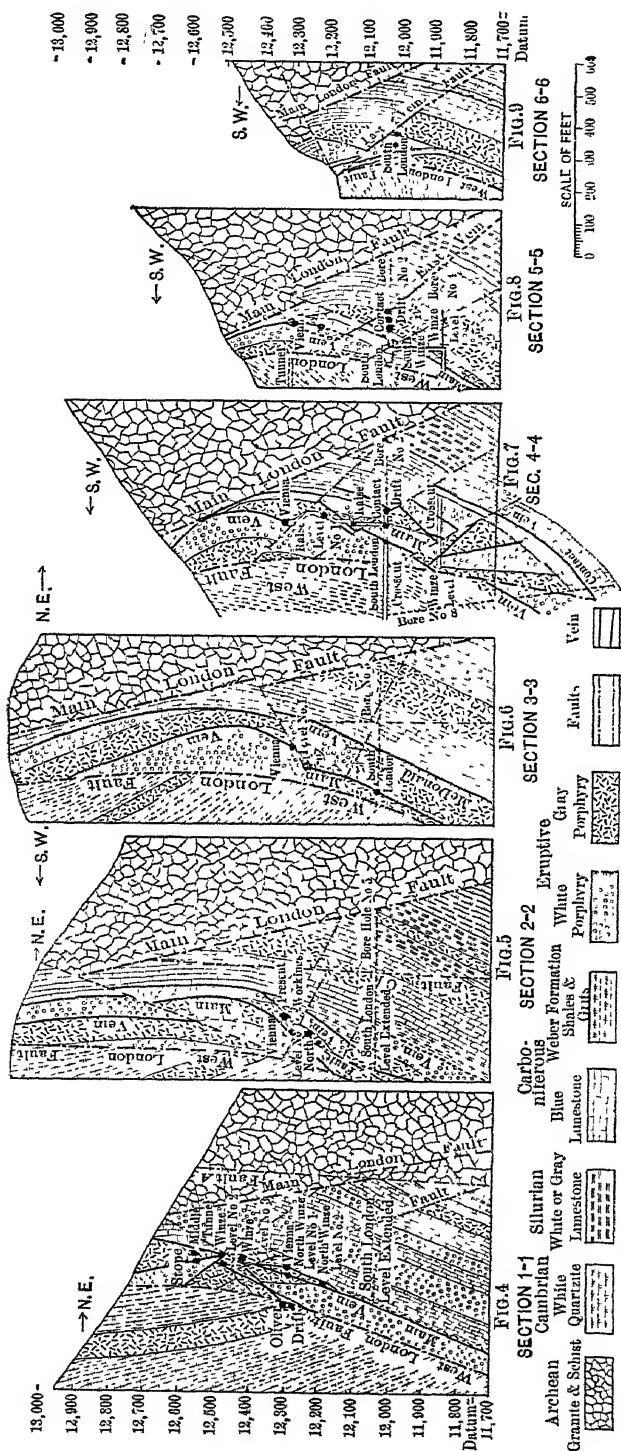


FIG. 3.—LONGITUDINAL SECTION ON THE MAIN VEIN, LONDON MINE. VERTICAL PROJECTION.



FIGS. 4 TO 9.—SECTIONS THROUGH THE LONDON MINE ON THE NUMBERED LINES OF FIGS. 2 AND 3.

The reconstruction of the process by which the present structural conditions were attained is shown in Fig. 1, and appears to be as follows:

(1) The uplift of the Mosquito range by which the sedimentaries and intercalated eruptives were arched into an immense anticline, finally relieved by the Mosquito fault with a break of 4,200 ft., leaving all the beds overlying the Archæan with an eastern inclination of 15° to 20° ;

(2) The formation of a great S-fold with a small anticline to the west;

(3) The relief of the stresses in this fold by the three London faults above described;

(4) Erosion to the surface of to-day. Fig. 1, which illustrates these facts, is platted accurately according to the surface-contours, and represents a section through the country from east to west, looking north and passing over the center of London mountain.

The London fault-system, though of such magnitude, did not relieve all the stresses in the ground, so two others occurred of later age, the first NE-SW. cross-faults lettered *A* to *I* in Figs. 2 and 3, pitching steeply southeasterly, and the latest with strike generally parallel to the London system, pitching at a low angle eastward. These last gave final relief and stability to the ground. Horizontal displacements in the latest series do not exceed 30 ft., while those of the cross-faults reach 60 ft. There is very little evidence of lateral movement in the London faults.

Ore-Deposits.

The ore-deposits in the mine are of two kinds: (1) replacement of the blue limestone at its contact with eruptives, and (2) true fissure-veins in the eruptives filled with quartz. The main London vein, which has been worked continuously for 20 years and from which the chief production of the property has come, is one of the latter, and all the workings between cross-faults *B* and *I* (except cross-cuts), are upon this vein.

Replacement-Deposits.—These originate in the planes of contact between limestone and the intruded or overlying porphyries, and are best developed in the northern part of the mine where the Blanchard cross-cut tunnel intersects two at 401 ft. and 535 ft. from its mouth. The first of these has a course N.

40° W. and dip 57° westerly conformable to the stratification, and is developed for 200 ft. in length. This vein is 5 ft. wide, and the ore siliceous oxide, copper-stained, but as the grade was not of commercial value, no further development was made and no cross-cuts run to test this vein elsewhere in the mine.

The other of these contact-veins, that lying stratigraphically at the top of the blue limestone, has been developed considerably in the northern portion of the mine and opened at numerous places to the southward by cross-cuts; the small raises from South London level are made on this vein (see Fig. 3, south of fault *I*, marked "stopes on contact"). In the upper and oldest portion of the mine, the "middle tunnel" workings, most of the stopes are upon this vein; this working was probably the only one visible at the time of Professor Emmons's examination, in which, however, he classified the limestone visible as "white" or Silurian instead of "blue" or Carboniferous, which is now proved.

The tenor of the ore in this vein was sufficient to justify mining in the northern portion, but in the central portion of the mine, where the main fissure-vein was most productive, this contact yielded no payable ore wherever opened by cross-cuts.

This contact is the horizon of the Leadville first contact, which has produced so largely in that district, an argentiferous galena, blende, and pyrite, oxidized in its upper part to cerussite, smithsonite, calamine, and limonite.

In the southern part of the mine, south of cross-fault *I*, this contact carries payable ore, but in both northern and southern portions it is evidently enriched by later solutions from the channel of the main fissure-vein. The dip is conformable throughout to the stratification of the limestone and thus proves the porphyry to have originally entered the sedimentary beds either as flow-sheets or laccolites. The eruptive intrusion was evidently a part of that throughout the Leadville district and synchronous therewith, older than all faulting, which is also true of the contact ore-deposits.

The dislocation of this contact by faulting as traced throughout the mine is interesting. In the southern end of the mine it is first reached by the South London cross-cut tunnel at 945 ft. from the entrance, and a drift run northward upon it for 210 ft. until cross-fault *I* is met. Up to this point the con-

tact produced good ore, semi-oxidized, assaying up to 9 oz. of gold per ton, but fragmentary, and the limestone overlies or forms the hanging-wall while the gray porphyry is the foot-wall, the dip varying from 34° to 60° eastward. North of cross-fault *I* the contact-vein and the limestone strata dip 54° to 77° westerly and the contact is reached by cross-cuts in the South London, Level No. 1 north, and Vienna levels at three places in each level. In all these places the contact dips westerly from 60° to 75° except in the McDonald stope between faults *B* and *C* in the northern portion of the mine, where it stands vertical. The ore from McDonald stope ran 7 oz. of gold and 10 oz. of silver per ton and was oxidized. To the southward all ore in this contact is sulphide except in the top of the stopes.

Fissure-Veins.—The chief ore-production of the property has been from the vein designated on the illustrative drawings as the "Main Vein." This is a well-defined fissure coursing N. 37° W. parallel to the faults and approximately also to the contact, and is opened for 2,980 ft. in length on the Vienna level. It is located between the main and West London faults and dips 65° westerly with remarkable uniformity from the Vienna level downward.

Unfortunately, the mill-holes and raises into the stopes on this vein are nearly all filled with waste or caved and inaccessible, so I was able to reach the back of the stopes only in three places and thus confirm the statement of the Superintendent that throughout the mine the dip of the vein at the top was eastward. While this is true, it was found to be due to the fissure in its upward course reaching the overlying limestone-porphry contact and coalescing therewith to the surface. Tracing this vein downward, it reaches the West London fault at an angle so acute that the fault and vein occupy the same fissure (see Figs. 7 and 8). The line of intersection of this vein with the fault is traceable throughout the workings for 2,000 ft. (see Fig. 3). As the fragmentary ore in the fault-fissure extends some distance below the intersection or junction with the vein, the stopes are continued accordingly, as shown in Fig. 3. Fragmentary ore is also found in the West London fault-fissure for 170 ft. south from cross-fault *H* to cross-fault *I*, where this fault dislocates the vein eastward into the contact-vein, as before described.

Diamond-drill boring is now in progress west of the West London fault to ascertain whether the main vein is found to cross the fault-fissure in depth; also whether the contact-deposits contain payable ore where lying horizontally in the syncline west of the faults.

The record of the first completed bore of the present series, No. 8, is just to hand and is platted on Fig. 7. The bore is 670 ft. deep and penetrates the blue limestone for 84 ft. after passing through a sulphide contact-vein 3 ft. thick overlain by 53 ft. of black shale, the lowest member of the Middle Carboniferous series of grits and shales. At 376 ft. depth a sulphide vein 1 ft. thick is crossed in a contact of gray and white porphyry; this is probably the main London vein on the west side of the West London fault. The further disclosures of these bore-holes are looked for with much interest.

Physical characteristics of the main vein: width, from 3 to 12 ft.; pay-streak, from 4 in. to 5 ft., average 18 in. wide, occurs chiefly on the hanging-wall but sometimes also on the foot-wall. The ore breaks clean against the wall, with frequent slickensides but seldom any selvage. There are very few spurs, and all are short and contain no payable ore. Very little evidence of movement or faulting in the fissure. The gangue is pure quartz with calc-spar and barite in small quantity and irregularly distributed. Occasional banded structure for short distances only, and the central cavities always lined with dog-tooth spar.

The smelter-returns show remarkable uniformity of values for periods of many months; for example, the shipments for the year 1908 vary only from 2.02 oz. of gold per ton as a minimum to 3.1 oz. maximum.

The production of the main vein from 1895 to 1910, inclusive, was 52,488 net tons, yielding \$2,018,920; an average net value of \$38.46 per ton after deducting hauling, freight, and treatment. These charges at the present time aggregate \$15 per ton; adding the average cost of breaking, sorting, and tramming to the bins, \$5, shows the total cost per ton = \$20.

The records do not show the ore properly credited to the different workings, but a fair division shows up to the year 1907, inclusive, that the stopes above the Vienna level produced 17,355 tons, yielding \$516,096, an average net value of \$26.26,

varying from \$20.63 in 1906 to \$40.77 in 1907; from 1905 to 1910, inclusive, the stopes between South London and Vienna levels, and that below South London level worked from the south winze, produced altogether 35,133 tons, yielding \$1,502,824, an average of \$42.77 net value per ton, varying from \$36.47 in 1908 to \$56.91 in 1905, indicating a decided increase of values downward, but with no visible evidence of secondary enrichment.

The total production of the property has been some \$5,000,000 gross, of which all but 10 per cent. has come from the main fissure-vein, the balance being from the contact-vein.

Ore is continuous throughout the entire length of the vein opened (2,980 ft.), and the greatest length of continuous stoping from 150 to 650 ft. in height is 1,850 ft.; ore is still continuous in all the ends and backs except where cut by faults.

All ore broken is hand-sorted in the stopes into two grades, the smelter-returns showing the relative proportions to have been 77 per cent. first-class, averaging 3.3 oz. of gold and 2.94 oz. of silver per ton; and 23 per cent. second-class, averaging 1.4 oz. of gold and 1.3 oz. of silver per ton.

The mean assay of 2,000 tons was 2.895 oz. of gold, 2.585 oz. of silver, 4.13 per cent. of lead, and the analysis derived from the smelter-returns was: Pb, 4.13; Zn, 2.66; Fe, 4.85; S, 6.11; SiO₂, 76.80; total, 94.55 per cent., the balance of 5.45 per cent. being probably calcium, barium, and carbon dioxide.

The other fissure-veins in the property are too small to justify development, although the tenor of the ore sometimes reaches 5 oz. of gold per ton.

Comparing the sedimentary strata penetrated by the horizontal bore-holes in the mine with those in the vertical hole 5,000 ft. west of the main London fault, the blue limestone is reduced in thickness from 200 ft. in the latter to 165 ft., 145 ft., and 163 ft. in three of the mine-holes, and the parting quartzite from 25 to 10 ft. This reduction is probably due to slipping of the more shaly beds out of place rather than to actual compression of the solid limestone and quartzite strata.

Thanks are due to W. K. Jewett, of Colorado Springs, Colo.; J. M. Kuhn, of Alma, Colo.; and R. H. Gummere, of South Bethlehem, Pa., officers of the London Mining & Reduction Co., for permission to publish this paper.

School Laboratory-Work: Sampling of an Ore Containing Coarse Gold.

BY CHARLES E LOCKE, BOSTON, MASS.

(New York Meeting, February, 1913)

THE little stamp-mill in the mining laboratory of the Massachusetts Institute of Technology and the work done by it have been well described by Prof. R. H. Richards and E. E. Bugbee in a paper read at the Albany meeting of the American Institute of Mining Engineers, February, 1903.¹

As run for student class-work the mill contains a feeder, stamps, plate, and vanner. Results in the past indicate that the work of these machines is just as efficient as that of a regular mill handling the same class of ore. It has been the custom to obtain ore principally from Nova Scotia, mainly because it is the nearest source of supply, and also because there is a considerable saving in freight-cost as compared with other districts. The gold in Nova Scotia ores is notoriously coarse, and for this reason it is almost impossible to obtain satisfactory results by sampling and fire-assay, unless an extremely large lot of ore is crushed and all of the contained gold pellets are saved. A very interesting and most extraordinary example of this fact has been presented by a recent run in the Technology stamp-mill.

Since the capacity of the mill is about 60 kg. per hr., and since in about 5-hr. run the students in any one section have an opportunity to make the necessary study of machines, it usually happens that several runs are made from one lot of ore.

When the shipment is received it is first dried and weighed, put through the Blake breaker, set at 1 or 1.5 in., and the crushed product is piled in a cone on the sampling-floor. When the crushing is completed the ore is usually stored away in barrels for future use; it is aimed to have each barrel of

¹ *Trans.*, xxxiv., 478 to 486 (1903).

the same quality as the others. The barrels are filled up by a man shoveling around the circumference of the cone, the first shovelful going to barrel No. 1, the second to No. 2, and so on. This method of sampling is not claimed to be absolutely perfect, but it is entirely satisfactory for the purpose in view. When the time arrives for a stamp-mill run two barrels are taken at haphazard from the lot and used, the ore being fed in 10-kg. lots, which are weighed out carefully by the students, and fed as needed into the hopper of the feeder. As each student weighs out a lot he takes a small grab-sample from the residue and places it one side into the sample-hod. When the end of the run is in sight, this sample-hod, which is usually nearly full of the sample, is carried to the crushing- and sampling-department, where the ore is first weighed, and then crushed in rolls until it passes a screen of the same size of aperture as that used in the stamp-mill. Any pellets of gold in the ore are flattened out by the rolls, and are readily picked out from the oversize of the screen. The undersize of the screen is further sampled down to an assay-sample, put through a 100-mesh screen, the pellets, if any, being saved, and the pulp assayed. The remainder of the material passing through the roll-screen is weighed out in 10 kg. lots, and fed with the coarse ore into the stamp-mill.

The sampling and assaying of a Nova Scotia gold-ore by the above method is entirely inaccurate, and the only justification for the procedure is that the results when compared with the true gold-content of the ore, as shown by the recovery of gold from the amalgams, and from the assays and weights of the products, will serve as a warning to the student for his future practice.

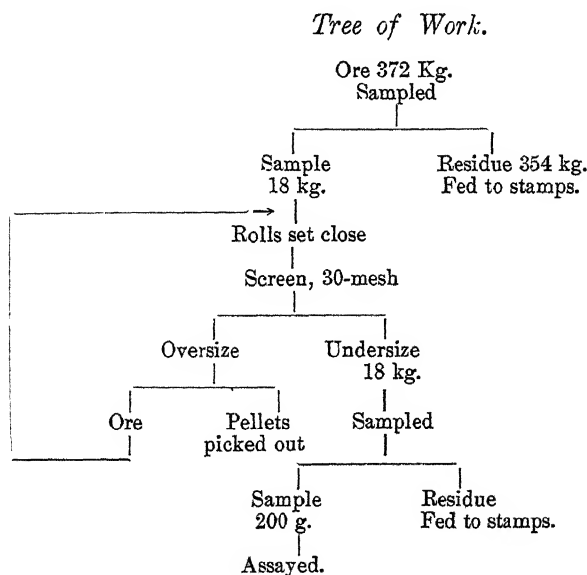
Usually the fire-assay runs lower than the mill-assay, and this seems perfectly natural when one considers that there are comparatively few nuggets of gold in the two barrels of ore, and the chances are that the students will miss them in the sampling. Occasionally, however, chance favors the sample, and it will contain more than the proper proportion of nuggets, and in that case the fire-assay rises higher than the mill-assay.

To present specific data: a load of ore received more than a year ago from Tangier, N. S., weighed 2,256 kg., net. Up

to the present time there have been three separate runs on this ore, as follows :

	Run No. 1. 402 Kg	Run No. 2. 353 Kg.	Run No. 3. 372 Kg.
Saved by amalgamation, per cent.....	95.58	97.32	77.02
Saved in concentrates, per cent.....	1.47	0.99	3.39
Lost, per cent.....	2.95	1.69	19.59
Concentration, tons, 100 into.....	2.47	2.29	2.27
Assay of concentrates, gold, oz. per ton.....	0.21	0.16	0.30
Assay of tailings, gold, oz. per ton.....	0.01	0.01	0.04
Mill-assay of ore, gold, oz. per ton.....	0.413	0.379	0.83
Fire-assay of sample, gold, oz. per ton.....	0.88	0.22	13.04

Looking over the tabulated data the first two runs are not greatly different from what is usually obtained. The mill-assays agree fairly closely, 0.413 and 0.379 oz. respectively. The first run had more than its proportion of pellets in the sample, and therefore the fire-assay shows higher. In the second run the reverse is true. It is in the third run, however, that is found such a remarkable example of the workings of chance. The following tree shows the procedure in the third run :



During the sampling it was noticed that only two pellets appeared on the 30-mesh screen, but these were both of good

size. No pellets were found between 30- and 100-mesh. The pulp through 100-mesh assayed 0.08 oz. of gold per ton. After the run the amalgams were cleaned, retorted, and the fine gold determined, also all the concentrates and tailings were dried, weighed and assayed, and the gold-contents figured. In this way the total gold in the original lot of 372 kg., including also the weight of gold in the two pellets, was obtained, and from these data was calculated the total gold there would be in a ton of ore, or, as it is called, a mill-assay. It is clear that the pellets should be included in this total, because if it had not been for the sampling they would have gone into the stamp-mill and would have appeared in the clean-up of the mortar.

The weight of fine gold in the two pellets was 8.0446 g., this being obtained from the 18-kg. sample. The amount of gold in the amalgams and in the products was 1.9614 g., which represents the gold in the 354 kg. that was fed directly to the stamps, and also the gold in the 18 kg. of undersize that went to the stamps. Computing this as ounces per ton, the mill-assay given below is obtained. Similarly, the fire-assay in ounces per ton based on sample weighing 18 kg., pellets weighing 8.0446 g., and pulp assaying 0.08 oz. per ton, is obtained.

Mill-Assay of Total Ore.

	Oz Per Ton.
Gold calculated from amalgams and products.....	0.20
Gold calculated from pellets....	0.63
Total.....	0.83

Fire-Assay of 18-Kg. Sample.

	Oz. Per Ton.
Pulp through 100-mesh.....	0.08
Pellets, add.....	12.96
Total.....	13.04

These two pellets of the third run are also accountable for the great drop in the saving by amalgamation to nearly 20 per cent. below the first two runs, and for the increase in loss from 2.95 and 1.69 respectively up to 19.59 per cent. What actually happened was that the ore fed to the stamp-mill in the third run really assayed only 0.20 oz. of gold per ton, whereas, in the first two runs it assayed twice that amount. If the two

pellets had gone into the stamp-mill they would have appeared in the clean-up of the mortar, and the following results would have been obtained—not greatly different, as far as extractions and losses go, from the first two runs :

	Run No 3 372 Kg.
Saved by amalgamation, per cent..... ..	94.48
Saved in concentration, per cent.	0.81
Lost, per cent..... ..	4.71
Concentration, tons, 100 into..... ..	2.27
Assay of concentrates, oz. per ton..	0.30
Assay of tailings, oz. per ton... ..	0.04
Mill-assay of ore, oz. per ton..... ..	0.83
Fire-assay, oz. per ton..... ..	13.04

Manifestly, the only way to sample an ore of this kind accurately is to crush the whole of it down to a comparatively fine size; but if this were done, then the stamp-mill run would have to be sacrificed. It seems proper, therefore, to sacrifice the accuracy in sampling in order to have a satisfactory stamp-mill run.

That the fire-assay is unreliable is well known to Nova Scotia miners, because any one in that Province unless he is a "ten-derfoot" will never place much confidence in the value of a mine on the testimony of fire-assays alone, but prefers always to have the results of mill-runs on several lots of ore.

The Gay-Lussac Method of Silver Determination.*

BY FREDERIC P. DEWEY,† WASHINGTON, D. C.

(New York Meeting, February, 1913.)

THIS old and well-known method of determining silver is, in bullion work, so far superior to the furnace-assay that it is looked upon with reverential awe by many, if not by most, users, and its ease of execution, with proper equipment, commends it highly where much commercial bullion work on silver is required.

The method is so thoroughly well known that a description of it here may seem to be unnecessary, but many thousands of determinations are annually made by it in the U. S. Mint Service and its practice is there reduced to an art. This is necessary both for the sake of economy of time, and because high-grade work with it requires constant practice. It is useless for a person who uses it only occasionally to expect to attain high accuracy with it, without spending more time upon a determination than is warranted in a busy commercial laboratory.

It is always employed in the Mint service whenever possible, but its chief field is in the determination of silver in standard silver (ingots and coin), which is 900 fine in silver and 100 fine in copper, and I shall first briefly outline its use on this metal and then take up various points in detail, especially as related to the accuracy of the results obtained.

For this determination 1,115 mg. of standard metal are weighed and transferred to a glass-stoppered bottle, the metal is dissolved in nitric acid, and 100 cc. of a standard solution of NaCl run in. The bottle is then vigorously shaken and a measured portion of a decimal salt solution added, and again shaken, if necessary. These operations are repeated until the silver is precipitated.

Standard metal being 900 fine, 1,115 mg. should carry 1,003.5 mg. of silver, which is a convenient figure to work with.

* Presented by permission of the Director of the Mint, and published jointly in the *Journal of Industrial and Engineering Chemistry*.

† Assayer, Bureau of the Mint.

Legally, standard metal may vary 3 in fineness above or below 900. Practically, however, it seldom runs belows 898 or above 901. Therefore the variation in actual silver present above or below 1,003.5 mg. is not excessive. For testing the standard salt solution we weigh up 1,004 mg. of proof silver. For convenience and accuracy we have single weights of 1,115 mg. and 1,004 mg.

The bottles should be of perfectly white glass, carefully made and well annealed; 8 oz. is a convenient size often used; the bottles are most conveniently handled in a circular frame or basket holding 10. For dissolving the silver, both the amount and the strength of nitric acid used may vary considerably without apparent effect upon the accuracy of the results, 8 cc. at 1.30 specific gravity to 25 cc. at 1.20 specific gravity being allowable. Some operators heat the bottle to remove nitrous fumes, others do not.

The standard salt solution is designed to have 100 cc. precipitate exactly 1 g. or 1,000 mg. of silver, but it seldom "shows" this exact strength. The word "shows" is used instead of "is" because the equivalent of the solution depends upon other factors besides its composition.

In general, two proofs should be used in every set of 10 bottles, unless several sets are to be run in rapid succession, when one proof in each set may answer. Some operators run an independent proof occasionally during the day and omit the proofs in the sets, but this proceeding is objectionable.

The next step in the method is the addition of 100 cc. of the standard salt solution (often improperly called "normal" salt solution). This is a very simple operation, but it requires the utmost care and constant attention to details if high accuracy is desired in the results. Only a minute variation in the amount of solution added will make a serious variation in the fineness of silver shown.

The Stas pipette is the one universally used in the Mint service. It is a simple pipette open at both ends and securely mounted on a wall bracket. The upper end is drawn out to a fine opening and is provided with a collar-cup to catch the drip. The lower end is comparatively large and must have a free and smooth discharge. The lower end is connected through a removable rubber tube, provided with a pinch-cock, with an elevated tank containing the salt solution. No stop-cocks, floats,

or graduations of any kind can be used on or in the pipette if rapid work is to be done. It is also questionable if any such arrangement can be as accurate as the simple filling of the Stas pipette and letting its contents run directly into the silver bottle.

In operating the Stas pipette the rubber tubing is slipped over the lower end and the pinch-cock opened. As soon as the solution comes out of the upper end of the tube, it is closed by the first finger of the left hand and the pinch-cock closed. The operator must now be sure that there are no air bubbles in the pipette. If such should appear they must be allowed to collect at the top of the pipette; the pinch-cock must be opened, the finger momentarily removed from the upper end of the pipette, and the pinch-cock closed again. When the pipette is full of solution and the pinch-cock closed, the rubber tube is withdrawn from the lower end of the pipette. This end must now be carefully examined to see that there is no surplus solution adhering to it or that the air has not commenced to ascend the tube.

If the lower end of the tube is in proper condition the silver bottle is now placed directly under it, the finger removed from the upper end and the solution allowed to flow into the bottle. The solution should flow out rapidly in a smooth, solid stream. Just as soon as the flow stops, the bottle must be removed from under the pipette. It is absolutely fatal to accuracy to attempt any adjustment of the drip of the pipette. There should be only just easy clearance between the bottom of the pipette and the top of the bottle.

The pipette is supposed to contain 100 cc. and on various accounts it is desirable that it should be fairly accurate, but it is not at all necessary that it should be absolutely accurate. The absolutely essential point about it is that it should deliver exactly the same amount of solution to each one of the 10 bottles composing a set. The amount delivered may be a trifle more or a trifle less than 100 cc., or it may vary slightly from a hot day in summer to a cold day in winter, but it should not vary between the first and last bottles of a set. It is a good plan to fill and empty the pipette a few times before beginning to fill the set. After withdrawing the bottle from under the pipette the stopper is dipped in distilled water and inserted into the neck of the bottle with care.

Having filled the set, the carefully stoppered bottles are

placed in a shaking machine and agitated for from 3 to 5 min. to settle the precipitate. The bottles are next placed upon a black shelf, technically called a "board," with a black background about as high as the shoulder of the bottle and about 3 in. back of the bottle, the whole being installed in a window, preferably with a northern exposure.

Up to this point in the method, the procedure is substantially the same in all the laboratories, but from here on there are slight differences in the manipulations. In common with many volumetric methods there is difficulty in this one in determining the end-point. In many descriptions of the method the operator is directed to add decimal salt solution in small measured amounts, with agitation between the additions, until no more precipitate is formed on adding the salt solution. Too much salt has now been added, and this excess must be determined by the addition of small measured amounts of decimal silver solution, and the amount of silver present in the metal determined by balancing these amounts.

This method is open to two serious objections. When exactly the proper amount of salt has been added to precipitate the silver present, a condition of equilibrium in the solution results, which is disturbed by the addition of either reagent with the separation of a precipitate. This obscures the end-reaction. These alternate dosings and shakings consume too much time for rapid work, and, after many shakings, the solution does not clear well.

In the Mint service, therefore, the operation known as "reading the cloud" has been substituted for these alternate dosings of the solution. This operation is far more rapid, but it requires a great deal of skill and constant practice to yield the best results. In reading the cloud, after the addition of the 100 cc. of standard salt solution and shaking, a measured amount, 1 or 0.5 cc., of decimal salt solution is added to each bottle. The delivery end of the pipette is placed against the neck of the bottle as far down as possible and the solution allowed to flow gently down the side of the bottle so that it will remain on the surface of the solution in the bottle with the minimum amount of mixing; by a slight rotary motion of the hand the decimal solution is then mixed with the upper portion (about $\frac{1}{2}$) of the bottle solution. This produces a cloud of AgCl in the solution, and the next step is based upon the

appearance of this cloud. Here the skill and visual condition, together with the personal equation, of the operator are of the utmost importance.

If the cloud be very heavy, two or more portions of the decimal salt solution are added, the amount depending upon the density of the cloud, and the bottles shaken in the machine again. If the cloud is light, only one dose of decimal is used. If the cloud is very light the bottle is again shaken by hand to bring more of the solution into reaction and the cloud again examined. As the result of this treatment one dose of decimal may be used, or the bottle may be shaken by hand again to bring the balance of the solution above the precipitate into the reaction. Here again a dose of decimal may be used, or the final reading of the cloud may take place. In the final reading the operator estimates from the density of the cloud what portion of the dose of decimal solution was consumed in precipitating the silver. Many operators estimate to 0.25 cc., others claim to be able to estimate to 0.1 cc. The results of the investigation given beyond indicate that on standard metal estimating to 0.1 cc. is not profitable under present conditions.

All the bottles are eventually brought down to the final cloud, and at the end the amount of decimal solution added to each bottle is recorded. The records of the assays are then compared with the proofs and the fineness of the samples determined. The actual fineness of a sample is shown entirely by the amount of decimal solution used as compared with the amount of decimal required by the proof. It is entirely independent of the amount of standard solution used. Therefore in determining the fineness of a sample we simply compare the amount of decimal used with the amount used in the proof, making allowance for the 0.5 mg. more of silver in the proof above the silver in 1,115 mg. of metal at the exact standard of 900, and taking into consideration the weight of the sample used. For instance, if the sample required 0.5 cc. less than the proof the sample would be reported at 900. This is not strictly exact because 0.25 cc. of decimal salt solution equals 0.25 mg. of silver, but 0.25 mg. is only 0.22 fine on 1,115 mg. and 0.5 cc. would be only 0.44 fine. However, this variation is too slight for practical consideration, and in general the finenesses are read directly from the difference between the sample and proof. Generally, also, the differences are read in 0.25 of a cc.

Practically we deduct $\frac{2}{4}$ from the amount of decimal solution used on the proof, and call it standard. Then for each quarter's difference from standard we add or subtract, as the case may be, from 900 in accordance with the following tabulation:

For $\frac{1}{4}$	0.2	For $\frac{6}{4}$	1.3
For $\frac{2}{4}$	0.4	For $\frac{7}{4}$	1.5
For $\frac{3}{4}$	0.7	For $\frac{8}{4}$	1.8
For $\frac{4}{4}$	0.9	For $\frac{9}{4}$	2.0
For $\frac{5}{4}$	1.1	For $\frac{10}{4}$	2.2

A second method of determining the end-point, which appears to be even more exact, but which requires more time, consists in adding only 0.25 cc. of the decimal solution after machine shaking, then estimating the number of quarters the solution will stand. These are added and the bottles shaken in the machine. This is continued until the quarter added after shaking produces no precipitate, in which case the last quarter is not counted, or else such a slight precipitate is produced that the quarter is counted but the bottle is not shaken again. The reading of the results is the same as before described.

In both these methods great care is exercised to avoid the addition of so much salt as to require back-titration with decimal silver. Back-titration is regarded with much disfavor.

In the regular work of the Mint Bureau it became desirable to ascertain just how much reliance could be placed upon the results obtained by the Gay-Lussac method as ordinarily executed in the service and an extensive investigation was undertaken for this purpose.

Six samples representing three melts of ingots were distributed among four laboratories in the service without informing any of the institutions just what the samples were. Table I. summarizes the results reported.

Notwithstanding the great preponderance of results from 898.9 to 899.1 shown here, the ingot melts were reported by the assayer of the mint where they were made as follows:

No. 1.....	898.5	fine in silver.
No. 2.....	898.25	fine in silver.
No. 3.....	898.5	fine in silver.

Table I. was submitted to the various assayers for criticism and comment and suggestions for improvements in the method.

TABLE I.—*Assays of Three Ingot Melts.*

Fineness.	No 1.		No. 2.		No. 3	
	A.	B.	A.	B.	A.	B.
898.3	1
898.4	2	1	2	2	1
898.6	2	4	2	2	3	2
898.7	1
898.8	2	1
898.9	5	5	5	5	6	6
899.0	2
899.1	10	9	8	7	8	7
899.2	1	2	1
899.3	1	1
899.4	1	1
899.5	1	1
899.6	2
Total.....	20	21	21	21	21	19

Five reasons were advanced to explain, in part at least, the differences shown, as follows :

1. Difference in samples taken for assay.
2. Non-homogeneity of the metal.
3. Working on a sample on different days.
4. The personal equation of the operator.
5. Accidents.

Two assayers suggested that perhaps under the actual, every-day working conditions of our service the method is not so accurate as it is supposed to be.

It was quite impossible at this time to deal with the personal equation of the operator or the chapter of accidents, but, in order to meet the first two of these reasons, a small bar of standard silver was prepared in the Bureau laboratory with especial care to avoid segregation of the metal. It was then rolled out and cut into small oblongs of which 30 to 35 were required to weigh 1,115 mg. To meet the third reason, four samples of the oblongs, each weighing approximately 3.75 g., sufficient for three determinations of the silver, were sent to each one of the same four laboratories at one time. Subsequently one laboratory ran two more samples. In all 66 assays were made and practically all of the bar was used for these assays. Table II. summarizes the results reported.

TABLE II.—*Assays of 22 Samples of a Small Bar of Standard Silver.*

3 assays showed.....	899.1	} fine in silver.
6 assays showed.....	899.3	
6 assays showed.....	899.5	
10 assays showed.....	899.6	
1 assay showed.....	899.7	
4 assays showed.....	899.8	
7 assays showed.....	899.9	
14 assays showed.....	900.0	
2 assays showed.....	900.1	
4 assays showed.....	900.2	
8 assays showed.....	900.3	
1 assay showed.....	900.4	

Table II. was also submitted to the four assayers for comment and criticism, and the investigation was continued upon six sets of ingot samples and six special samples. These samples were sent out to the four institutions for comparative test assays, in a long series. In no case was any institution informed as to the source of the metal, or given the results of any previous assays. The samples were sent out in lots of four in all cases but one. In several cases all four samples in a lot were the same, and in two cases this fact was stated in sending them out. In many cases two or three samples in a lot would be the same, but this fact was not disclosed. In one case six samples representing the three regular samples of a melt of ingots were sent out with this statement, but it was not shown which were the duplicate samples.

In the ordinary operations of melting ingots and taking samples from the melts, it is quite possible that there might be some slight difference in the samples themselves. Tables III. and IV., however, indicate that in the present condition of the method such differences would be masked by differences in the results due to the method itself.

The regular samples from a melt of ingots are generally too small to allow a sufficient number of test assays to be made. From three melts one mint supplied larger samples and they were sent to the four laboratories for test assays, and Table III. summarizes the results reported, together with the original assays made at the mint of origin of the samples.

TABLE III.—*Assays of Three Melts of Ingots.*

Melt No. 132.						
Fineness.	Original Assays.		Test Assays.			
	Mint of Origin.		Mint of Origin		Independent.	
	A.	B.	A.	B.	A.	B.
898.3	2	1	1	1
898.4	2
898.5	1
898.6	1	2	3	4
898.7
898.8
898.9	3	9
899.0
899.1	1	1	3	1	2	2
899.2
899.3	3
899.4
899.5	1	1	2	2	2
Total.....	2	2	6	6	13	22
Report.....	899.3					47

Melt No. 133.						
898.3	1	2	1
898.4	3	1
898.5	2	1
898.6	1	6	2	8	5
898.7
898.8
898.9	3	4
899.0	1
899.1	1	1	2	4
899.2
899.3	1
899.4
899.5	1
900.0	1
Total.....	2	2	6	6	18	18
Report.....	899.1					48

Melt No. 138.						
898.6	9	2
898.7
898.8	1
898.9	4	7
899.0	2	2
899.1	1	6	6	2	5
899.2	1
899.3
899.4
899.5	1	2
Total.....	2	2	6	6	18	17
Report.....	899.5					47

This table indicates that there was a decided tendency at this mint to report their ingots high.

Another mint supplied two sets of ingot samples that were sent out to the four laboratories in the general series of samples, and Table IV. summarizes the results reported and gives the original assays made at the mint.

TABLE IV.—*Assays of Ingot Samples.*

Melt No. 175.										
Fineness.	Original Assays.			Test Assays.						Total.
	Mint of Origin			Mint of Origin.			Independent.			
	A.	B.	C.	A.	B.	C.	A.	B.	C.	
898.4	1	1
898.5
898.6	1	1
898.7
898.8
898.9	5	1	6
899.0	1	1
899.1	1	2	2	1	5
899.2	1	1
899.3	1	1	1	3	1	3	2	9
899.4
899.5	1	8	7	1	17
899.6	1	1	3	3
899.7	2	1	2	5
899.8	3	3	4	10
899.9	2	2
900.0	1	5	6
900.1
900.2
900.3
900.4	2	2	4
Total.....	2	2	2	6	6	6	18	17	18	71
Report	899.3									

Melt No. 176.										
898.9	3	3	3	9
899.0
899.1	1	3	1	5
899.2
899.3	2	1	1	2
899.4
899.5	6	2	11	19
899.6	2	2	1	2	1	2	6
899.7	2	3	1	6
899.8	2	1	1	4
899.9	1	1	2
900.0	5	11	2	18
900.1
900.2	1	1
Total.....	2	2	2	6	6	6	18	18	18	72
Report.....	899.5									

A third set of samples from the second mint was sent out in a set of six and the instructions informed that they represented a melt of ingots, but the duplicates were not indicated; Table V. summarizes the results reported together with the original mint assays.

TABLE V.—*Samples of an Ingot Melt Assayed at One Time.*

Melt No. 180.									
Fineness.	Original Assays.			Test Assays.					
	Mint of Origin.			Mint of Origin.			Independent.		
	A.	B.	C.	A.	B.	C.	A.	B.	C.
899.3	1	1	1
899.4	1	2
899.5	6	5	9
899.6	2	1	1	1
899.7	1	1
899.8	2	1	4	5	4
899.9	3	2
900.0	6	4	5	3	2	2
900.1	1	1
Total.....	2	2	2	6	6	6	18	18	18
Report.....	889.5								
									72

The general tendency of the original reports on these three melts appears to be a trifle low, and this is supported by the low test assays on Melt No. 176, made at the mint of origin.

In Tables III., IV., and V. all the results reported have been given. Undoubtedly, some of these were vitiated by accidents. These tables also indicate that there is a general laboratory equation similar to the personal equation of an operator.

In order to eliminate differences in the samples as a source of difference in the assay results, one mint prepared two special samples, in regard to which the assayer wrote: "I have prepared two samples of standard silver (6 oz. each). I used coin-ingot melted a number of times with stirring; finally cast in a closed mold that had been chilled with ice; poured silver at as low temperature as possible. Resulting bars were very free from oxidation. They were, however, thoroughly cleaned, and rolled into strips and cut."

The samples cut into oblongs, of which it took about 30 to weigh 1,115 mg., were forwarded to the Bureau, where they were further mixed and portions of 3.75 g. weighed out into

small envelopes, giving ample material for three determinations of silver in each envelope. Ten envelopes were sent at various times to each one of the four institutions. Each laboratory made 30 determinations of silver on each of the larger samples, making a total of 120 assays on each, but in tabulating the results reported a certain amount of discretion has been exercised. In general, results at either extreme in fineness have not been included in the table unless they were reported at least twice by two laboratories, or five times by one laboratory; Table VI. summarizes the results together with the original assays at the mint of origin.

TABLE VI.—*Assays of Two Special Bars.*

Bar No. 1.

Fineness.	Original Assays.	Test Assays				Total.
	Mint of Origin.	Mint of Origin.	Lab. No. 1.	Lab. No. 2.	Lab. No. 3.	
898.4	6	3	9
898.5	2	2
898.6	10	18	11	19	20	68
898.7	8	8
898.8	1	1
898.9	6	11	17
899.0
899.1	2	6	8
Eliminated	3	4	7
Total	10	30	30	30	30	120

Bar No. 2.

898.4	1	5	4	9
898.5	4	4
898.6	5	14	12	14	12	52
898.7	2	2
898.8	3	3
898.9	4	7	1	13	21
899.0	1	1
899.1	3	15	18
Eliminated	4	3	3	10
Total	10	30	30	30	30	120

Another mint (Laboratory No. 3 above) prepared two special granulation samples, in regard to which the assayer wrote: "I am submitting two samples of 6 oz. each of the most homogeneous standard silver that I have been able to prepare. These samples were prepared as follows: Standard silver ingot

granulations assaying from 899.1 to 899.8 and weighing in the aggregate a little over 30 oz. were melted with frequent stirring and the entire melt granulated. Two samples of about 8 oz. each were then segregated and 18 assays made from the different portions of the sample. 17 of these assays went 899.5."

These samples were divided into portions weighing 3.75 g. each at the Bureau. It was not noticed at first that they were really only one sample, so that 11 envelopes of No. 1 and 9 envelopes of No. 2 were sent to each one of the four institutions. Table VII. summarizes the results reported, together with the original assays at the mint of origin.

TABLE VII.—*Assays of Special Granulations.*

Portion No. 1.						
Fineness.	Original Assays.	Test Assays				Total.
	Mint of Origin	Mint of Origin.	Lab. No. 1.	Lab. No. 2	Lab No 3	
898.6	8	8	1	17
898.7	6	6
898.8	5	5
898.9	3	13	13	29
899.0	3	3
899.1	10	1	12	12	35
899.2	2	2
899.3	2	6	8
899.4
899.5	17	23	1	1	25
Eliminated	1	2	2
Total.....	18	33	33	33	33	132

Portion No. 2.						
898.6	2	11	5	18
898.7	5	5
898.8	7	7
898.9	3	7	12	22
899.0
899.1	9	15	14	38
899.2
899.3	1	1
899.4
899.5	17	16	1?	17
Eliminated	1
Total.....	18	27	27	27	27	108

In this case Laboratory No. 3 is the one that supplied the special bar samples. The laboratory supplying these special granulation samples is the same one that forwarded ingot

samples Nos. 132-3-8 and the same tendency to obtain high results is again shown here.

More than 15 years ago, in the article entitled, *The Actual Accuracy of Chemical Analysis*,¹ in speaking of the "degree of accuracy exhibited in actual every-day practice" I said: "In estimating this, little weight will be given to the evidence afforded by the agreement of duplicate or multiple determinations by the same chemist; for I am convinced that such agreement is a delusion and a snare." The results reported on these granulation samples afford a striking illustration of the soundness of this position. This metal was assayed 257 times. The laboratory preparing it reported it 56 times at 899.5 fine, but the other laboratories reported this figure only 3 times and one of these determinations was questioned by the laboratory reporting it. On the other hand, the work of the other laboratories shows that this metal could not have been over 899 fine.

Another small bar was prepared in the Bureau laboratory with especial care, and cut up into oblongs requiring from 30 to 35 to weigh 1,115 mg. These oblongs were thoroughly mixed and small samples of 3.75 g. put into envelopes. Ten of these envelopes were sent to each one of the four institutions, and Table VIII. summarizes the results reported.

TABLE VIII.—*Assays Bureau Special Sample No. 2.*

Fineness.	Lab No. 1.	Lab. No. 2.	Lab. No. 3.	Lab. No 4.	Total
899.1	6	6
899.2
899.3	6	3	9
899.4
899.5	5	6	11
899.6	4	4
899.7	12	12
899.8	9	9
899.9	6	6
900.0	2	13	9	17	41
900.1
900.2	5	3	8
900.3
900.4	1?	13	14
Total...	30	30	30	30	120

The laboratories here are numbered the same as in Table VII., and No. 4 is the mint of Table VII.

¹ *Trans.*, xvi., 370 (1896); *Journal of the American Chemical Society*, vol. xviii., No. 9, p. 808 (Sept., 1896).

A final sample was made up at the Bureau by uniting the oblongs left over from the regular coin work of the Bureau, which were of widely varying finenesses, and, without any special mixing, weighing up samples of 3.75 g. Ten envelopes were sent to each one of the institutions and Table IX. summarizes the results reported. As this sample had no claim to uniformity all the results reported are given.

TABLE IX.—*Assays of Bureau Coin Scrap Sample.*

Fineness	Lab. No 1	Lab. No 2.	Lab No 3	Lab. No. 4	Total
898.6	4	4
898.7
898.8
898.9	7	2	9
899.0	1	1
899.1	1	12	8	21
899.2	4	4
899.3	6	6	12
899.4	2	2
899.5	9	12	6	20	47
899.6	6	6
899.7	3	3
899.8	1	4	5
899.9
900.0	2	2
Lost...	4	4
Total...	30	30	30	30	120

The laboratory numbers here are the same as in Table VIII.

As a final test of the method when practically applied to ingot work, three large samples were taken from each one of nine melts of ingots and assayed in various laboratories. Each sample was assayed in duplicate five times and in triplicate once, making a total of 39 determinations on each melt. Table X. summarizes the results reported.

As a general conclusion from the elaborate tests herein given it may be stated that two operators, working upon identical samples of standard silver and making four determinations each, may differ as much as 1 fine in their reports. Having thus established the capacity of the method as at present carried on as a commercial operation, attention is now being given to improving the method so as to reduce the allowable limit of difference.

It may seem unnecessary to many of my readers, but my experience shows that there is much confusion and uncertainty

TABLE X.—*Assays of Nine Coin Ingot Melts.*

Fine- ness.	Melts.									Total.
	1.	2	3	4	5	6	7.	8.	9.	
897.7	1	1
898.0	1	1	1	3
898.2	1	2	1	4
898.4	1	1
898.6	7	5	7	3	1	23
898.7	1	1
898.8	3	3	1	7
898.9	7	1	3	1	3	15
899.0	2	2	1	5
899.1	12	4	10	15	10	7	2	3	5	63
899.2	2	2
899.3	1	4	4	6	7	2	8	32
899.4	2	5	7
899.5	5	8	8	13	14	15	16	19	17	115
899.6	2	1	4	3	5	15
899.7	3	3	3	1	10
899.8	1	1	4	1	1	8
900.0	7	1	3	1	5	2	8	27
900.1	1	4	5
900.4	2	2
Total..	39	39	39	39	39	39	39	39	39	351

in stating the composition of precious metal bullion. This is generally done in parts per thousand or fineness. It is similar to stating composition by percentage, but in bullion work there is much misunderstanding regarding the decimal point. Intelligent people who would never think of expressing fifty per cent. by .50 per cent. habitually write five hundred fine as .500 fine and incorrectly use the decimal point before the numerals in stating fineness. Whenever "fine" or "fineness" or an equivalent expression is used, in stating the composition of bullion, the decimal point should not be used unless the figures following it express a quantity of less than 1 fine. Again, some people incorrectly use a preliminary "0" when the fineness is less than 100. They would not think of writing 05 per cent. but habitually do write 052 fine.

Again, some people mistake fineness as being the exact equivalent of percentage, but it is only one-tenth of percentage, and in considering the accuracy here shown by the Gay-Lussac method it should be noted that 1 fine is the equivalent of 0.1 per cent.

Biographical Notice of Floris Osmond.

BY ALBERT SAUVEUR, CAMBRIDGE, MASS.*

(New York Meeting, February, 1913)

FLORIS OSMOND, Honorary Member of the American Institute of Mining Engineers, born in Paris, March 10, 1849, died at Saint-Leu near that city, June 18, 1912. Taken suddenly ill with congestion of the brain while on his way from Paris to his home, he reached it in time to write on the typewriter his last wishes, one of which was that no biographical notice of his life should be published. Holding Osmond's memory sacred, as those who have known him will, is it our duty to conform with his desire? Shall no mention be made in our *Transactions* of the beautiful life and great achievements of one of our most honored members? Is it not, on the contrary, incumbent upon us to disregard Osmond's instruction in this matter and to place on record our admiration for the nobility of his life, for his talents and for the importance of his work, that younger men may be inspired by it with a desire to be guided by worthy motives in the path of unselfish pursuits? I cannot feel that in following this course I am disloyal to Osmond's memory and I am strengthened in this belief by the opinion of men of high honor whom I have consulted before undertaking the writing of this obituary notice. One of them, Professor Henry M. Howe, justifies my decision better than I could do it myself when he writes :

"Let us look frankly at the matter. Here we have a most extraordinary life, a life of rare beauty and power of inspiration to the rest of mankind. In my opinion we should not deprive mankind of the stimulating influence of the example of M. Osmond simply because he shrank from publicity. Our young men see every day how greed and vulgar push succeed, and with what material rewards they are crowned. They see too rarely the more valuable crown which comes to self devotion. A permanent and public recognition of the beautiful qualities of Osmond's life cannot fail to appeal to the imagination of young men and to help to deflect them toward leading a noble rather than a sordid life. Therefore I feel that the *Revue de Métallurgie* and other bodies owe it to the public to give prominence to his career by means of biographical notices."

* Professor of Metallurgy and Metallography at Harvard University.

The following short account of Osmond's career is taken for the most part from a sketch which I wrote for the *Metallographist* in 1900.

Osmond graduated from the École Centrale des Arts et Manufactures, where he studied metallurgy under the late Professor Jordan. After a few months spent in the machine shops of Fives-Lille, M. Osmond entered the employ of the Blast Furnaces, Forges & Rolling Mills Co. of Denain and Anzin, which was then adding to its works some Bessemer converters and open-hearth furnaces; he assisted in starting the steel-making department, and, although devoting his time more particularly to the open-hearth furnaces and to the laboratory, he familiarized himself with all the details of a large metallurgical establishment. In 1880, M. Osmond joined the metallurgical staff of the famous Creusot works (Schneider & Co., Le Creusot, France) and there met M. Werth, with whose collaboration he wrote his first, and now classic, contribution to the application of the microscope to the study of iron and steel, entitled, *Structure Cellulaire de l'Acier Fondu*. Drawn together by a similarity of taste, Osmond and Werth sought, along scientific lines, the solution of the daily problems coming to their notice in the actual running of the works, and so often remaining unsolved. In 1884, Osmond left the steel-works to devote himself exclusively to the study of scientific metallurgy; and it was chiefly at the private laboratory installed in his Paris apartment that he conducted those admirable researches which have rendered his name famous on two continents, although some of his work was performed at the Sorbonne in the laboratories of Troost and of Moissan. His second classic paper, in which he sets forth the allotropic theory of iron, was written in 1887, being entitled *Transformations du Fer et du Carbone dans les Fers, les Aciers et les Fontes Blanches*. In 1888, the French Society for the Encouragement of National Industry awarded to M. Osmond a prize of 3,000 francs and in 1898 a second prize of 2,000 francs. In 1895, the same society conferred upon him the Lavoisier medal¹ in recognition of his masterly presentation and successful defense of the allotropic theory of

¹ The Lavoisier medal is awarded every six years to the author, whatever his nationality, whose work has most contributed to the progress of French industry. The illustrious Sainte-Claire Deville was the first recipient.

iron. In 1898, the American Institute of Mining Engineers elected M. Osmond an honorary member. The Russian government, in 1906, acting on the recommendation of Tschernoff, bestowed upon him the decoration of the Order of St. Anne, second class, a very high distinction. The same year he received the Bessemer medal from the Iron and Steel Institute, the President, Sir Robert Hadfield, alluding to him, on that occasion, as "a great Frenchman who represented that wonderful French accuracy and devotion to research which they would do well to imitate in their own country to a greater degree, and who was one of the leading authorities in the world on the diagnosis of steel and its peculiarities."

When we realize the rare talents of Osmond, his devotion to his work and his splendid achievements, it must be a source of surprise that he was not a member of the French Academy of Sciences. He would have been admitted long ago, writes Le Chatelier, but his candidacy could not be considered; for never would he have consented to make the indispensable calls and solicitations. Like Béranger, his proud nature would not permit him to play the part of the solicitor; and of him also it may be said by the academicians as was said of Molière: "*Rien ne manquait à sa gloire; il manquait à la nôtre.*" Only a short time before Osmond's death Professor Le Chatelier had prepared a report of his scientific work, in support of his nomination as Corresponding Member of the Academy—no solicitations being in that case required. Unfortunately Osmond was a graduate of the *École Centrale*, while the majority of the members of the Academy claim the *École Polytechnique* or the *Université* as their *alma mater*. The nomination was indefinitely postponed. Professor Le Chatelier's report was never read. The pity of it all! And still more incomprehensible than the fact that he was not elected to the Academy is the fact that he was never decorated with the ribbon of the Legion of Honor! This omission passes understanding.

To Osmond we owe the important discoveries, that metals frequently combine to form definite chemical compounds, and that these compounds frequently form solid solutions. Thus he lifted the veil which hid from our view the constitution of metallic alloys. Before him, vague and conflicting opinions were entertained as to the nature of alloys, while the existence

of solid solutions was unsuspected, and very few chemical compounds of metals were known. The subject had not received the attention which its importance demanded; but Osmond's researches were the starting-point of great scientific activity and the progress made by following his metallographic and thermal methods has been rapid and fundamentally significant.

Osmond discovered the thermal critical points of iron and founded upon their existence the allotropic theory of that metal, a theory equaled by few for brilliancy of conception. The allotropic theory of the hardening of steel, a natural sequence of the allotropy of iron, gave us the first rational and scientifically acceptable explanation of that unique phenomenon, which before him had baffled the keenest metallurgists, or had evoked untenable and often silly explanations.

Osmond's metallographic and thermal studies of iron and steel have added abundantly to our knowledge of these metals. To him we owe the discovery of the non-magnetic solid solution of iron and carbon, austenite (existing above the thermal critical range), and the transition-constituents, martensite, troostite, and sorbite, marking as many distinct and important steps in the transformation, on cooling, of the solid solution austenite into the ferrite-cementite aggregate. The reality of some of these transition-forms was at first doubted; but their existence has been repeatedly confirmed, and their importance as constituents of commercially treated steel has increased. Martensitic, troostitic, and sorbitic steel are at present industrial products of well-defined characteristics.

To the technology of metallography, Osmond's contributions have been numerous and valuable; no one in this field has been a surer guide, and no one has had more followers—and to follow was to admire him.

He devoted much attention to the investigation of various iron-alloys, especially to those of iron and nickel. His discovery of the existence of irreversible iron-nickel alloys was the starting-point of investigations, carried on by many workers, which have resulted in the intimate knowledge we now have of the constitution and properties of these important products.

The constitution of "high-speed steel" never ceased to interest him, and he wrote about it a number of valuable

articles. Without detracting in the least from the just fame of the inventors, it may be affirmed that Osmond contributed, indirectly at least, to the epochal discovery of high-speed steel.

His work bears the stamp of a master-mind, thoroughly equipped for original scientific investigation. Profound erudition, logic, and brilliancy are among the qualities which must impress the student of his writings. He possessed in a high degree the fearlessness, thoroughness, and caution so necessary for successful research, while his modesty, generosity, and unselfish devotion to his work place him on a high ethical level.

Osmond was a poet, an appreciative and discriminating admirer of the various manifestations of the beautiful in art, in literature, in science, in civic and domestic life. His translation into French verse of the Latin poem, *Ferrum*, written during the eighteenth century by the Jesuit, Xavier de la Sante, is a little master-piece. In the October, 1912, issue of *La Revue de Métallurgie* Le Chatelier publishes a poem by Osmond, entitled *Profession de Foi*, which furnishes proof, both of the author's talent as a "versificateur," and of the loftiness of his views and aspirations. During the last years of his life, he devoted much time to interior decorations, with pencil and paint, of the walls of his house at Saint-Leu.

As is well known, Osmond had been deaf for many years. His delicately organized nature suffered so keenly from this infirmity that he spent the last years of his life as a recluse, receiving only occasional visits from his intimate friends, and declining the most urgent and loving requests to mix with the outside world. He left nearly 500,000 francs, the bulk of his fortune, to philanthropic or semi-public institutions.

Few scientists have been held in warmer esteem by workers of all countries, even by those who did not always share his views. On the occasion of his death Professor Le Chatelier, as editor of the *Revue de Métallurgie*, received many letters of condolence, from some of which the following brief remarks are extracted:

"I have just learned during a short stay in London the sad news of M. Osmond's death. I take the liberty of expressing to you my sorrow at the great loss experienced by France and international science through the death of so eminent a savant. You know how much I was personally attached to M. Osmond whose kind interest contributed much to the success of my metallographic studies."

—Carl Benedicks.

"It is with great regret that I have learned that the founder of the modern theories of the iron-carbon alloys. M. Osmond, had passed away." —F. Wust.

"We agree with you and with your French colleagues in appreciating as they should the services rendered by the deceased to the development of scientific metallurgy and metallography. Although the deceased through an exaggerated modesty has expressed the desire that no eulogy be published concerning his talents or his work, we believe we are not violating his wish in calling the attention of our readers, by a short notice, to the loss suffered by international science through the death of M. Osmond." —Editorial Staff, *Stahl und Eisen*

"It is with the greatest sorrow that I have learned of the death of M. Osmond, our dear friend, whose memory cannot perish. I pray for him." —D. Tschernoff.

"The death of Osmond is a loss for the scientific world and will be felt in all civilized countries as keenly as in his native land. All his writings and letters reflect the tranquility of soul of a luminous mind which knew how to blend the most amiable modesty with the sublime sweep of its thoughts." —E. Heyn.

"M. Osmond was one of the glories of your beautiful France from which I carried away the most sincere affection." —N. Belaiew.

"The death of your countryman, M. Osmond, has deeply grieved me, and I have the honor to express in the name of the Nobel Committee for Chemistry our deep regrets on the occasion of the great loss that your country and science have suffered by the death of that illustrious savant."

—Olaf Hammarsten, President Committee Nobel for Chemistry.

"It is indeed with great grief that I have heard the sad news about the metallurgist whose name was a household word throughout the world.

"His personal kindness to me over a long period of time I shall ever remember with gratitude and respect. I am sending you a telegram of condolence, and may I ask you to convey my expression of sympathy to M. Osmond's relatives.

"Perhaps a little later on something may be done to perpetuate his name by way of a memorial. I need hardly say how much I should like to be allowed to join in such an expression." —R. A. Hadfield.

"His death will be mourned by all concerned in the metallurgy and metallography of steel, but Osmond's own work will be the most enduring monument of his fame. Both as a metallurgist who has profited by Osmond's work, and as a personal friend who has felt the charm of his personality, I desire to express my sense of the loss we have sustained" —Walter Rosenhain.

"I have always regarded Osmond as the greatest pioneer in science, and at a metallographical meeting of the Iron and Steel Institute many years ago, described him as the prince of metallography." —J. E. Stead.

"I am greatly obliged by your letter of yesterday, informing me of M. Osmond's wishes that no biographical notice concerning his work and life should be published. That desire is certainly characteristic of the true modesty of M. Osmond's great genius, and undoubtedly his friends and admirers will wish to respect it. Nevertheless, we feel that we cannot omit the publication of a few lines in the Journal of the Institute in appreciation of his remarkable qualities, in doing which I understand we are following the same course which you propose to take in the *Revue de Métallurgie*." —G. C. Lloyd.

"As I think you know, British scientific metallurgists held the work of Osmond in great esteem; and on behalf of the University of Sheffield, and of the Iron and Steel Department of the Royal School of Mines, I beg to convey to you profound regret at the death of your distinguished friend." —J. O. Arnold.

"His death leaves a void in science, and especially in metallurgy, that it will be impossible to fill; they have, in fact, suffered an irreparable loss.

"I need hardly say how much his work was appreciated by us; and words fail to express the esteem and regard in which he was held in this country.

"The whole world is poorer now that he has passed away from us."

—W. Gowland.

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Valuation of Iron-Mines.

BY JAMES R. FINLAY, NEW YORK, N. Y.

(New York Meeting, February, 1913.)

At first blush one is tempted to say that iron-mines are like any other mines, and that principles governing mining-property in general will apply to them. But there are certain considerations which seem to repay reflection. Gold and copper are more or less independent of transportation. The proof of this is that those metals are profitably mined and shipped from the remotest corners of the earth. Their value is determined by the cost of production on the ground. This is quite true of gold, not quite so true of copper, not at all true of iron. If New York were a fishing-village and Chicago a swamp, Calumet & Hecla might still be a bonanza. The main element of its value is the 60 lb. of fine copper per ton which it produced for years. But the Mesabi Range is absolutely dependent upon the industrial activity in the United States. Iron-manufacture in Europe would give it no value whatever. If Chicago and New York had a population of Peruvians or Hindus, the Hull-Rust mine would be as worthless a bit of landscape as any tundra in Alaska. I have discussed these facts more or less casually and dimly in my book on the cost of mining and in my report to the Tax Commissioners of Michigan.¹ I shall try here to bring them out in more detail.

I have already referred to some differences in the problem of mining as between the more precious metals, *e.g.*, gold, and the less precious and more abundant minerals, like iron and coal; but it is interesting to note that the literature of the eco-

¹ In 1911 I was engaged by the State Board of Tax Commissioners for the State of Michigan to appraise all the mines of that State for taxation purposes. I placed a valuation of \$69,815,000 on the copper-mines, assuming that the price of copper would average 14 cents a pound. Using an average price for iron-ore, about equal to the 1911 schedule of prices, I appraised the iron-mines at \$119,500,000 on an expected tonnage of 196,354,883 tons.

nomics of mining has come mainly from men who have been thinking of gold. Our present ideas of mine-valuations were mainly originated by H. C. Hoover, whose point of view was almost strictly at that time that of an English investor in gold-mines. Hoover's ideas, or their equivalent, were first used extensively in America by J. Parke Channing and others in working out a value for the "porphyry coppers." From thence they have spread to a more or less general acceptance.

The cardinal principle of Hoover's system of valuation is simply that the value of a mine is a capitalization of future profits. Given the margin of profit in an ore, the amount of ore, and the time required to get the profit, the value is merely that profit as it will appear in a series of dividends discounted back from the future date of payment. I most heartily subscribe to this principle and believe it to be the only one worth considering; but, like every other general principle, it is only the starting-point for many complex deductions. For example, an ore which is not marketable at all to-day may be highly in demand 20 years from now. It would be foolish to apply Hoover's principle to support an argument that because an ore has no present market it has no present value.

The fact is that the conception of profits depends on :

1. The value of an ore; that is, its average price.
2. The average cost of getting it out and getting it to final market.
3. The amount of ore and the rate of getting it out.

Now, when we consider how these factors apply to different kinds of mines, we find that with gold-mines the first factor presents no uncertainty whatever. Gold is not only marketable at a fixed price, but in unlimited amounts. The gold-miner does not have to consider either how much of his product he can sell or what he will get for it. Inasmuch as his gold commands a fixed market, so his gold-ore is always in the market. He immediately transfers his attention to the quality and quantity of it. Now, if gold were not hard to find, gold-ore would not have an unlimited market. Therefore, the prime uncertainty is the ore itself. In spite of the fact that development is carried on as fast as possible in the average gold-mine, the amount of ore is usually the principal subject of uncertainty. Once mining is successfully established, the cost is

practically determined; so that almost the whole matter of valuation is the life of the mine.

Now, if we turn our attention to iron-ores, we find ourselves confronted with different circumstances. Omitting discussions of properties in the prospecting or developing stage, which have little or no influence in the markets of the world, we find that iron-ores of good grade are known in immense quantities running up into billions of tons. In the large, therefore, the question is not primarily the supply of ore. The main questions are decidedly the price at which it can be sold and the amount that can be marketed each year. In valuing iron-mines in comparison with gold- or copper-mines, we transfer our chief uncertainties into quite a different channel.

I have asserted that in iron-mines an error of 10 per cent. in estimating the average price of an ore will often make far more difference in the valuation than any possible mistake as to the life of the mine. I do not wish to minimize or to ignore the life factor either, because in the case of many iron-properties the amount of ore yet to be discovered in the ground will be very great. Nevertheless, in almost every mine of importance, the uncertainty as to the ultimate production of ore is confined between certain limits. What I wish to insist upon is, that with most of such mines the difference in valuation caused by using the minimum estimate of ore or the maximum estimate of ore upon the property will not be so great as the difference produced by a variation of 10 per cent. in the price to be received for that ore at the blast-furnace.

This fact has been quite clearly recognized in discussions of the valuations which I made in my report to the Tax Commissioners of Michigan. Almost immediately after this report was published, the price of iron-ore declined some 10 or 12 per cent. below the average I had counted upon. During the past year I have come to see the crucial importance of this question of price and have given it much attention. I have tried to obtain some logical idea of the basic factors which will determine the price of iron-ore in a given field. In the hope that I have made some progress in this direction, I submit the argument which will follow.

Before proceeding with the discussion it seems worth while to point out that computations of an average price for the fu-

ture, based upon the price for the past, must in themselves be unstable and in a measure contradictory according to the number of years of past experience taken to establish the average. There have been periods of very high prices and other periods of very low prices. A five-year average will be one figure and a twenty-year average will be another figure. After all, the selection of a price, even in the light of experience, if not actually a matter of guess-work, must be a matter for the exercise of judgment. It is not a thing which is mathematically certain. It is not at all impossible that the situation might warrant the expectation of a price quite out of range of past experience. There might be reasons for believing that iron-ore would have a higher or lower price than ever known before.

Over against this let us try to work out on other grounds an idea of the limits within which prices may fluctuate under the pressure of competition. For convenience and to get a concrete illustration to argue about, let us have in mind the Lake Superior mines.

The factors which I will take to be vital for any iron-ore field are the following:

1. Ores that are pure enough to be inherently desirable for iron-smelting and manufacture.
2. A great volume of such ores, sufficient to warrant the great investment of capital in the steel business. This naturally demands a firm foundation in assured ore-supply.
3. The assurance that such ores can be supplied within the industrial field to which they belong at least as cheaply as ores from any other source.

I propose to point out as definitely as possible how the Lake mines meet these requirements. The first thing to do is to define the field they supply.

Lake Superior iron-ore goes into an industrial field of the first magnitude. It is rivaled only by a similar field in Europe which comprises roughly the middle third of Great Britain and a tract on the Continent covering Belgium, northern France and western Germany. The American field is a rough ellipse, the axis of which runs from Philadelphia to Chicago and whose sides are Lake Erie on the north and the south boundary of Pennsylvania on the south. The major axis is near the path described by the American center of population in its progress

westward. That center has reached a point in Indiana, not very far southeast of Chicago. Pittsburg is about the middle point of the whole field. A moment's reflection will show that this area is precisely that area which has the closest contact with the main portion of the population of the United States and Canada; that since the densest part of the population is east of the center of population, so the most active industrial area is naturally east of it; that the area is best situated geographically for economical distribution to the bulk of consuming demand, for all kinds of manufacture. Lake Superior iron-ore enters this region from the north and there meets its necessary complement, the great fuel-supply from the Appalachian coal-field, which lies just to the south.

It is little wonder that this area has become preponderant in iron-manufacture; that it accounts for from 80 to 90 per cent. of the whole iron business of the country.

Now, I invite attention to this inference: If Lake Superior can meet any competition as far east as Pittsburg, it is bound to supply half of the iron used in this field. If it can meet outside competition all the way to the coast, it is bound to supply nearly the whole demand. I propose to argue on this basis. I propose to leave out of account the extraneous or gratuitous factors of taxes, royalties, and profits, which are merely consequences allowed by the natural advantages which these ores enjoy, and to consider only the fundamental factors of:

1. Cost of mining.

2. Distances of transportation (*a*) by land, and (*b*) by water, of these Lake ores against any known or likely rivals.

Present metallurgical practice seems to be to treat the Old Range² ores as an accessory to Mesabi ores, using them as a

² The Lake Superior iron-ore region is divided into six "ranges," or districts: three in Minnesota, the Vermillion, Mesabi, and Cuyuna ranges; and three in Michigan, the Gogebic, Marquette, and Menominee. Some other fields in Canada and elsewhere are as yet unimportant. The Mesabi Range presents marked differences from the others both geologically and commercially and in importance far outweighs all the other ranges combined. It is situated in northern Minnesota and consists of a gigantic sheet or bed of rich iron-formation containing in its unaltered state from 30 to 35 per cent. of metallic iron, the remainder being principally silica. This bed is from 500 to more than 1,200 ft. thick, lies nearly flat and is very gently folded, so that the outcrop on the surface is frequently two or three miles wide. Surficial alteration of the formation has in places removed the silica, leaving great bodies of nearly pure iron oxides containing approximately 60 per cent.

desirable mixture. The proportions are about 70 tons of Mesabi to 30 tons of Old Range. As near as I can make out, the average mining-cost of all these ores, omitting taxes, is about 60 cents a ton for Mesabi (largely steam-shovel) and about \$1.50 for the others (all underground). I am talking about complete costs, including everything except initial plant. This figures for the whole product 87 cents a ton. The average iron-content of these ores I believe to be about 52 per cent. natural. This means that the average mining-cost is $\frac{87}{52}$ —1.67 cents per unit.

Let us take this ore to Pittsburg as the central point of the manufacturing field. 70 per cent. has a land-haul of about 90 miles to lake, and 30 per cent. has a land-haul of 37 miles; the average is 68 miles. 70 per cent. has a lake-haul of 850 miles and 30 per cent. has a lake-haul of 650 miles; the average is 790 miles.

From Lake Erie to Pittsburg all ores have a land-haul of 160 miles.

Total, 228 miles haul over a level country, with a preponderance of down-hill pull for the loads.

All lake-haul is over the protected waters of inland lakes. Both factors are highly favorable. Land-haul appears to average at present about $\frac{66}{100}$ cent per ton-mile; lake-haul, about $\frac{9}{100}$ cent per ton-mile. These figures are accurate enough for practical purposes. Applying them, we get the following costs at Pittsburg:

of iron, when dried. The other ranges are similar masses of iron-formation which have been sharply folded, tilted, and intruded by igneous rocks. Their attitude and complexity render both exploration and mining more difficult. The whole Lake country is covered by a mantle of glacial débris and outcrops of actual ore have been very rare.

The Mesabi Range is pretty thoroughly explored and about 1,500,000,000 tons are developed. The Old Ranges are not so fully developed, in spite of being generally older. They have from 300,000,000 to 400,000,000 tons of ore developed. Annual shipments from the Mesabi are about 30,000,000 tons, from the Old Ranges from 12,000,000 to 15,000,000 tons. The Mesabi ores are less desirable for the blast-furnace because their fine texture makes a lot of flue-dust and impedes proper working. The Old Range ores are less friable and are sometimes very hard. These qualities give them a premium in the market over Mesabi ores of 25 cents a ton. For fuller geological descriptions see *Monograph LII., U. S. Geological Survey*, by C. R. Van Hise and C. K. Leith (1911).

For commercial ratings and prices see *Iron Ore Manual, Lake Superior District*, by Rukard Hurd. F. M. Catlin, sales agent, St. Paul, Minn.

	Cents Per Unit
Mining, 87 cents per ton.....	1.67
Land-haul, $228 \times 0.66 = \$1.50$ per ton.....	2.89
Lake-haul, $790 \times 0.09 = 0.71$ per ton.....	1.37
	<hr/> 5.93

Lake ores are sold on the basis of four standard grades, as follows :

	Nat. Iron (Undried). Per Cent.	Iron (Dry). Per Cent.	Phos. Per Cent	Moisture Per Cent.
Old Range Bessemer.....	55	61.12	0.045	10
Mesabi Bessemer.....	55	61.12	0.045	10
Old Range Non-Bessemer.....	11.50	58.52	12
Mesabi Non-Bessemer.....	51.50		12

For Bessemer ores a premium is allowed for phosphorus less than the standard.

For all ores allowances are made for iron-contents varying from the standard. Silica is also carefully taken into account. For a penalty, see Hurd's *Iron-Ore Manual*.

Ore is one-third Bessemer. The Bessemer quality is worth 50 cents a ton additional; say, for the average, 17 cents additional.....	0.33
Total base value.....	<hr/> 6.26

Now let us look at some competitors. Cuba can produce 1,000,000,000³ tons or more of iron-ore, which, when roasted, or "nodulized," will also contain 52 per cent. of iron. The ore is all low in phosphorus and contains 0.75 per cent. of nickel. These factors will, I believe, make it a formidable rival. I will count in the Bessemer differential, but because the nickel is not yet established as a premium factor, we will neglect that, except to consider the nickel as so much iron.

	Cents Per Unit.
Cost of mining, say, 27 cents per ton, 40 cents nodulized.....	0.75
Nodulizing, \$1.....	1.89
1,200 miles sea-haul at 0.09 = \$1.08 per ton.....	2.04
375 miles land-haul at 0.66 = \$2.70 per ton.....	5.09
Total cost.....	<hr/> 9.77
Value of Cuban—Lake Superior base cost.....	5 93
Add Bessemer quality—70 cents per ton.....	1.34
(I estimate Lake Superior Bessemer at 0.45 Phos. Cuban ore is much lower, 0.25, and for this commands a premium.)	
Total base value.....	<hr/> 7.27
Differential in favor of Lake Superior.....	2.50

³ For description of this interesting and unique field, see various papers in our *Transactions*, by Jennings S. Cox, Jr., C. K. Leith and W. J. Mead, Arthur C. Spencer, C. Willard Hayes, Willard L. Cumings and Benjamin L. Miller, Dwight E. Woodbridge, and James E. Little, in vol. xlii, and C. M. Weld, in vol. xl.

This is equal to \$1.30 per ton at Pittsburg.

The only other field which can figure in billions of tons is Brazil. Let us appraise its factors. The ore-fields are in the tropics in a mountainous country, 380 miles from the sea, with plenty of rain. Rail-haul will cost more than in the Lake Superior region, but let us ignore that. Let us take mining at any figure you please, say 25 cents a ton, and let us take the ore to be 62 per cent. of Bessemer natural.

	Cents Per Unit.
Its base value will be	5.93
Add Bessemer quality (Phos. 0.045).....	1.00
Total value.....	6.93

Its cost will be :

Mining, 25 cents per ton.....	0.40
Land-haul, 730 miles at 0.66—\$4.82 per ton.....	7.77
Sea-haul, 5,700 miles at 0.09—\$5.13 per ton.....	8.27
Total cost.....	16.44
	6.93
Differential in favor of Lake Superior.....	9.51

This is equal to about \$5 per ton of Lake Superior ore.

Now, there is bound to be a point where Lake Superior will meet both Cuba and Brazil on equal terms. This point will be, in terms of land-haul, that distance east of Pittsburg where its freight will equal half the differential in favor of the Lake Superior ores which exists at that point. In the case of Cuba that differential is \$1.30 per ton. One-half of that is 65 cents. In terms of freight at $\frac{6.6}{100}$ per ton-mile, this means 100 miles. Cuban ores, therefore, will meet Lake Superior ores at that point on even terms and east of that place they will have the advantage. Very roughly, this gives to Cuba about one-quarter of the manufacturing field. I see no reason why Cuba cannot establish for that proportion of the industrial field substantially the same position which the Mesabi Range has in the Lake Superior field; that is, it can furnish, say, 70 per cent. of the iron.

In the case of Brazil, the differential is equal to about 388 miles land-haul east of Pittsburg. Brazil cannot, therefore, compete even on tide-waters. At that point Cuba has a differential of 7.73 cents a unit, or \$4.10 per ton, over Brazil.

It may be argued that outside ores may be brought in with lower freight-rates than I have used. I answer that any such argument is a question of competition in transportation, not competition in iron-ores. It cannot be seriously urged that Americans can do better work on the Atlantic ocean than on the Lakes, or build better railroads in Brazil than in America. Whenever transportation has to compete, it will do so on its basic advantages. It is very possible indeed that estimates of transportation-cost much lower than those of our Lake region may prove to be practicable; but if they do, it only means that transportation-costs there are too high and may be reduced.

In this argument I have omitted two factors of competition which have some bearing. There is a duty of 12 cents a ton on Cuban ores. This is equal to an 18-mile land-haul. Our meeting-point, therefore, will have to be moved that much further east of Pittsburg.

Another factor is difference in initial plant-cost at the mines. I believe this factor to be in favor of Lake Superior, because the cost of a nodulizing-plant per annual ton is probably greater than that of the average Lake Superior mining-plant; but I do not think that this will make a difference of more than a few cents, probably 5 cents a ton. It must be remembered that docks, boats, railroads, etc., are transportation-factors, and I believe they are nearly identical for the two fields. If Atlantic boats cost more per carrying ton they can work 12 months a year, against 7 months on the Lakes.

In both fields some properties are at an advantage or disadvantage, as the case may be, compared with the grand average used in this comparison. In Cuba I have allowed a railroad-haul of 25 miles. Some properties will not have this. On the whole, I see nothing to vitiate the conclusion that Cuban ores have a basic advantage up to a point 100 miles east of Pittsburg without the tariff, and 125 miles east of Pittsburg with the tariff.

Population of the United States and West Indies where Cuban ores have the advantage, omitting Georgia and Alabama, but including Florida, is about 31,000,000.

Influences of These Facts Upon Prices.

It should be evident from the above that Lake Superior ores are firmly established in the greater part of our American iron

manufacturing field. The only possible competition that appears to be important is that from Cuba, and even that is only incipient. Even if Cuban ores do come in to displace Lake Superior ores in one-quarter of this field, the total effect on the Lake markets is likely to be inconsiderable. In the first place, the demand for iron and steel in America is expanding, and the need for ores is increasing at the rate of at least 25,000,000 tons each decade. The importation of from 8,000,000 to 10,000,000 tons a year of these Cuban ores will seem a bagatelle in the development of the next ten years, and they cannot be brought into the market at a rate fast enough to influence greatly the market. Besides this, it must be taken for granted that Cuban ores will not be marketed except at remunerative rates, and this fact will automatically protect Lake Superior owners in a large part of their profits, particularly in that great interior market which apparently must always remain exclusively their own.

It seems fair, therefore, to conclude that the differential in favor of the Lake Superior ores at Pittsburg, which seems to be a central and representative point for the whole field, is about \$1.30 per ton against the nearest competition; that in spite of any possible introduction of outside ores into the eastern part of the field, the total demand from the Lake Superior mines is sure to increase instead of diminish, and that even now the price of the Lake Superior ores is governed more by competition from within than by competition from without. The proof of this is that the actual profit from such ores is less than that warranted by their advantages over foreign competition at Pittsburg. In 1911 my conclusion was that the Michigan mines were fairly secure in a margin of \$1 a ton profit, including royalties. It appears that this profit is only 80 per cent. of what they might expect if they were to meet Cuban competition on even terms.

It has always seemed to me that the discovery of the Mesabi Range introduced into the Lake region a period of intense internal competition on a scale not likely to be repeated, and under circumstances not likely to be repeated. At that time the Mesabi was like a foreign field, rudely replacing the established producers, and making them fight for their lives. The Old Range mines had shipped in 1890 and 1892 nearly 10,000,000 tons a season; but in 1893 and 1894 the demand

from the whole field was only 7,000,000 tons a year. With a lessened demand, the supply was being increased by leaps and bounds by enormous discoveries of fine ore which could be mined by steam-shovels. Is it any wonder that iron-ore seemed ridiculously easy to get? These circumstances all concurred at a time of universal low prices. Every staple of commerce reached its record point of cheapness about 1894; after that time a rising scale of prices set in which has continued to the present day.

As soon as the disturbance produced by the appearance of the Mesabi Range could adjust itself, the price of Lake Superior iron-ores began to reflect more and more their natural advantages. I have believed that an estimate of their future price should be based upon their fluctuations since the time when the untoward conditions just described as not likely to be repeated, had disappeared. If we had another Mesabi on the

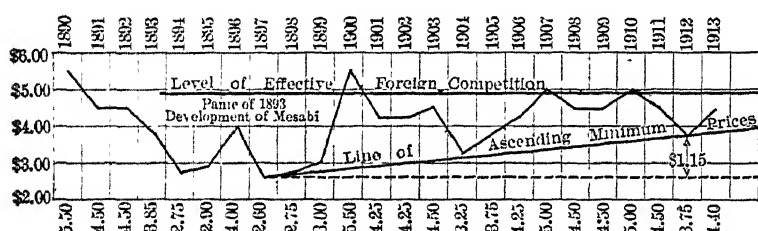


FIG. 1.—CHART OF IRON-ORE PRICES, SHOWING EFFECT OF DISCOVERY OF MESABI IRON-RANGE AND PANIC OF 1893. PRICES ARE FOR OLD RANGE BESSEMER ORE.

horizon I should try to take it into account; but as such a phenomenon appears to be less and less likely, I think it logical to believe that iron-ore prices will crowd more and more closely to the level allowed by grand-scale competition from foreign fields.

Actual Course of Prices.

As an interesting commentary upon these considerations let us take the course of prices since 1890. The Old Range Bessemer ores will serve this purpose, although it should be noted that these ores do not represent the average Lake ore considered in the comparisons given above. Fig. 1 shows the yearly prices at Cleveland.

The value of these ores at Pittsburg, reasoned out according to their basic factors in competition, I calculate as follows:

Iron value, 55 units at 5.93 cents.	\$3.26 per ton.
Bessemer quality.....50 per ton.
Premium for texture.....25 per ton.
Differential over Cuban ores.....	1.38 per ton.
<hr/>	
Total competitive value at Pittsburg.	\$5.39 per ton.
Less freight from Cleveland.....	1.00 per ton.
<hr/>	
Base value at Cleveland.. . . .	\$4.39 per ton.

Now, since the competition is based upon the bare operating-cost of Cuban ore, with no allowance for commissions, duty, amortization of nodulizing-plant, royalties, or profits, it is plain that this base price is not a commercial basis of competition, but a price at which Cuban competition will be impossible in the Pittsburg field. If we may believe that Cuban producers might be willing to assume all these costs for a margin of 50 cents a ton, which seems narrow enough, we are forced to conclude that the competitive level for this grade of ore at Cleveland is about \$4.90 per ton.

Now, the actual average of prices from 1897 to 1913, inclusive, is \$4.10 per ton. The average of lowest prices on the chart for the years 1897, 1904, and 1912 is \$3.20 per ton. But each succeeding cycle of low prices has not been so low as the preceding one. The low price of 1912, \$3.75, was \$1.15 per ton higher than the low record of \$2.60 in 1897. Furthermore, we see that the average price for the 17 years has been 90 cents a ton higher than the sloping base-level of lowest prices. It seems legitimate to believe that average prices have actually advanced \$1.15 per ton between 1897 and 1912; that in 1897 prices were fluctuating above a minimum of \$2.60 per ton and in 1912 above a minimum of \$3.75; that if the real average is 90 cents a ton above the low average of a period of years we might as reasonably calculate that \$4.65 is an average price for these ores in 1912 as that \$3.50 was an average during the late 90s. As a matter of fact, the average for the seven panic years, 1893 to 1899, inclusive, was only \$3.12 per ton. In 1911, I based my estimate of the ruling price for these ores on a seven-year period, 1904 to 1911, inclusive, and obtained a figure of \$4.50. I adopted this figure for my valuations, thinking that those years represented both good and bad commercial conditions covering about the same number of years before and after the panic of 1907. It has been hard to feel

that this was good reasoning, especially facing the fact that prices dropped after 1911. Nevertheless, an average obtained by using the seven years ending 1911 or 1912 would yield the same figure. The considerations enumerated in this article have fortified me considerably in the belief that the figure I adopted in 1911 was reasonable, after all. I now think that the logic of the situation points to a price of between \$4.50 and \$5 a ton for Old Range Bessemer at Cleveland, and that prices either lower or higher will only be temporary. These figures I believe are trustworthy as long as transportation-rates remain on their present basis. Foreign ores can only disturb the equilibrium by means of lower freight-rates. If such rates are put in force so as to permit important foreign competition the rates in the Lake country will come down. The mine-owners, at any rate, should be fairly protected in a margin of profit approximately equal to that allowed by the above prices and present rates.

The Value of Money.⁴

It has been urged that money employed in mining should be allowed a high rate of interest because of the extraordinary uncertainties of the business. While I do not wish to dismiss this contention cavalierly, it seems to me that about all there is to say on this point is that a valuation is logical only on the assumption that the factors used in making it are correct. If such factors are judiciously taken they are as likely to err on one side as on the other—as likely to exceed as to fall short, con-

⁴ For the benefit of readers who have not followed discussions of mine-valuations I will explain that the question of interest-rate in valuation is this: the mine being of limited life, the purchaser is entitled to get his money back, plus interest, during the life of the mine out of dividends or profits. Each dividend is supposed to be an installment of principal together with a sum for interest on his purchase money. The question of the rate of interest to be allowed makes a vast difference with a calculation of what that purchase price should be. For instance, suppose a mine can pay \$1,000,000 a year for 30 years, and then cease. If the purchaser demands 5 per cent. on his money he can pay \$14,740,000. If he demands 7 per cent. he would pay only \$11,380,000. At 10 per cent. he will pay only \$8,490,000. In terms of stock-valuation such a property would be worth:

At 5 per cent.....	\$100.00 a share.
At 7 per cent.....	76.50 a share.
At 10 per cent.....	57.60 a share.

Thus a higher rate of interest means a lower estimate.

sequently there is no need of counting on any further doubts than such as are inherent in all human enterprises. I have generally assumed that 5 per cent. was a normal interest- or discount-rate. If that is so it is a fair figure to use in a mine-valuation, which should be nothing but a candid inquiry into the present value of expected profits.

Merits of Mining-Properties.

At the risk of repeating some commonplaces I venture to add that mining-property has some exceedingly good points—for reasons so obvious that they are often lost sight of. In the nearly perpetual battle against absolute fakers, and those other fakers in a highly-respectable garb who exploit the public by inducing gross over-valuations of the shares of good mines, it is quite possible for conservative mining engineers to lay too much stress on the dangers and uncertainties of mining. These uncertainties lie very largely, almost wholly, in that stage of the business which is only preliminary; the prospecting stage, the search for a starting-point. When that stage is passed mining changes its character completely and instead of being one of the most hazardous, it becomes one of the least hazardous of businesses. It is true that this applies only to developed ore with an assured market, and that the business of getting out ores already discovered is often mixed with venturing money to find new deposits; but it is quite easy to exaggerate the uncertainty of finding ore. Not infrequently, reserves are measured by mere conventions which take no real measure of the amounts which can be safely relied upon. Furthermore, if ore is developed for from 10 to 15 years ahead, which is very frequent among iron-mines, the question of opening up more ore is not a question of prime financial importance. Take Lake Superior for a capital example. Every mining-man in that field knows that enough ore has been developed to maintain the present output for 30 or 40 years. It surely is more to the point to calculate what these supplies are worth than to be worrying about ores yet to be found. A great deal of money and a great deal of intelligence is expended each year in exploring new ground, but it is absurd to say that the business is hazardous because it is wise to make

such expenditures. The fact is that such properties are highly favored.

In the first place, they are indestructible. This advantage is too obvious to talk about. In the next place, these properties get their value from nature as well as from man; that is, they are partly independent of management. A Colorado operator once remarked that he had no use for a mine that would not stand bad management. There is truth in this dictum and every mining-man knows what he means. A mine's profits may be blighted by lax administration; it may be made ridiculous by improper financing, but if the mine is good it will make money with the most ordinary management. Its success does not depend wholly on the extraordinary talents of its owner. Commercial, financial, manufacturing, or transportation enterprises, on the other hand, may depend absolutely upon the ability of some one man. When that man dies or weakens the whole structure may go straight to the scrap-heap and the business may never earn a dollar. Not so with a great ore-body. It will be more valuable with a genius to exploit it, but it will pay good dividends for plain John Smith or Will Jones. To take a good example, I will venture to say that the average mining-man feels far more confidence in the United States Steel Corporation as a mining concern than as a manufacturing concern. At any rate, he feels that it is infinitely more stable and secure with its mines than it would be without them; that its mines are an asset guaranteed by nature itself.

If we approach the subject of mine-valuation without careful analysis of what we are talking about it is easy to confuse the risks of the business of prospecting with those of mining. In the first case there is no question of valuation. It is simply a question of spending money to find a mine and develop it if found. If the venture is successful the adventurer will get a large return, usually 20 to 30 per cent. a year in the case of Lake Superior mines, on the money actually embarked. His risk is that he will find no ore. He may try many times without finding it. Everybody knows that; and commercial practice makes the reward of success high. But when a man successfully finds a mine, develops it and makes it pay, he has a valuable property. It then becomes one of the surest of assets. It would then be absurd to value it as if it were still a pros-

pector's vision. If one did that the mine would be worth no more found than unfound. Quite the contrary, I think; the established mine is worth more than it seems to be worth. The owner is fully assured not only of what he has found, but generally that he can find more.

It is quite a privilege to own a great natural resource, whether that resource lies in the inherent value of a precious metal or whether it lies in an advantage over competitors through a favorable situation.

DISCUSSION.

CHAIRMAN JOHN BIRKINBINE:—Having been invited to preside at this session, the Chairman feels that the part he takes in discussions should be limited, although the subject presented is one of much interest to him and there are diverse viewpoints concerning methods of valuing expectancies from what may be considered "ore in sight," a term itself subject to various definitions.

Estimates of quantities upon which valuations are made may differ without subjecting the estimators to criticism of unfairness if made by a prospective purchaser or by a seller, or if they are used as the basis for taxation. The hopes of the seller may influence a liberal allowance for estimated reserves, while the expectancies of a willing purchaser are less optimistic, and the tax assessor is expected to limit his consideration to the producing capacity of a mine, with possibly modest allowance for reserves.

The knowledge that a property contains or yields mineral or the expectancy that it may be a producer of value adds to the estimate placed upon it by any of the above classes, and a just determination of this value is often a complex problem.

The paper under discussion favors determining mine-valuations upon an apparent profit per ton based upon the difference between the selling price of ore and the expense of mining and marketing it for a term of years or for the expected life of the mine. Such a method must be modified by allowances for changes in utilization which may favorably or unfavorably influence the demand in the future; the fluctuations in the selling price, increased cost due to deeper mining, advancing wage rate, unfavorable legislation, labor troubles, increasing transportation charges, insurance covering accident to the plant as

well as to workmen, etc.; all these may offset savings resulting from the employment of improved equipment and appliances for mining, beneficiating, or transporting the mineral.

The total probable output of a mine is an important factor, affected by the method applied or which may be advantageously used to secure the largest ultimate tonnage, while the location of a deposit in relation to a market for the quality of ore it produces and the facilities for reaching this market are controlling factors.

It is well known that some iron-ores considered desirable in former years are not now employed, while others which had been placed on waste dumps are now in demand, owing to changes in methods of manufacture and fabrication or advanced mining, handling, and transportation facilities. Some waste dumps have been reworked and mineral of value obtained therefrom.

The progressive character of mine management, or the reverse, may be properly considered as an asset or a discount in mine-valuation, for a property well exploited gives greater promise than one operated inefficiently or wastefully.

It is therefore evident that the valuation of any mine or mineral property, whatever its product, present or prospective, may be, must be treated individually, and the result secured by considering the existing reserves, the present and probable future quality of these, and the present and anticipated costs of winning the ore, as well as its future value.

A retrospect of the iron and steel industry shows such remarkable changes in the past quarter of a century that speculation as to the future may, with propriety, be limited, for the next few decades may develop such radical changes as to seriously influence estimates of value made from present knowledge.

HARRISON SOUDER, Cornwall, Pa.:—I did not come prepared to discuss the paper; in fact, I did not know that the paper had been written, and therefore did not have a chance to read it. It is a pretty important subject. It seems to me that each mine is a problem by itself and has to be considered in that light. Mr. Finlay has taken whole mining fields and figured out approximately the ore valuation for that particular district, but it seems to me that each mine manager knows that his

proposition is an individual one and has to be considered on its own merits.

There is one point I noticed here, and that is that Mr. Finlay seems to think that taxes do not count. Speaking of the mine which I manage, I know our taxes are going up pretty fast; in fact, there has been an increase of 50 per cent. in eight years, and it has added quite an item to the cost of the ore. It is part of the mining cost, as we figure it. Mr. Finlay's curves show, as near as I understand them, that the prices of ore are going up. The curves from our property for the last forty years show the general trend of prices downward. Also the mine is going down deeper, which means more equipment and more expensive machinery; the grade of ore is decreasing to a certain extent, the same as the Lake Superior ores are, and to put these ores on a level with some of the high-grade ores which are being discovered in different parts of the world means the installation of expensive machinery, and that adds to the cost of the ore per ton. All these points, to my mind, have a bearing on the valuation to be placed on any given ore property, so that you cannot take a whole ore field and say that the value of the ore in that field is so much a ton. You must take each property by itself.

The tendency nowadays is to tax you on your industry and enterprise in investigating your property. In our case we have put diamond drill holes down through our ore body to determine its character and extent, and spent much money on mining plant and equipment, and the fact that we have done this has caused the tax assessors to increase the assessed valuation on the mines to an unreasonable amount. I understand the same thing has been done in Michigan, where a great deal of the prospecting has been held up because, if you thereby find a little additional ore, you have to pay taxes on it for many years in advance of the time when it will be used. I do not think I have any more to say, but will be glad to hear from others.

J. L. W. BIRKINBINE, Philadelphia, Pa.:—I understand that the discussion is not to apply wholly to iron-ore, but may apply to coal-mines as well as other mines?

THE CHAIRMAN:—The title of the paper is "Valuation of Iron-Mines," but it seems to me to be the determination of

iron prices. The valuation of iron-mines is a basis upon which the writer apparently presented certain prices of ore, which he presumed, from the data at hand, will bring the Pittsburg region as a center into which the iron-ores of Lake Superior are expected to meet competition from Cuba and from Brazil. He might have mentioned the importations from the Scandinavian Peninsula, which are really more prominent with us than anything else, except Cuba, at the present time. He might have referred to other districts which are producing iron-ores.

The title of the paper leads one to believe that the discussion would be only pertinent upon the valuation of iron-ores, but, as the writer himself goes into the price of the iron, I feel the discussion should include not only the valuation of the mine but the valuation of the product and the influence that that product, and its valuation, has upon its utilization.

Mr. Vallat, can we hear from you on this subject?

B. W. VALLAT, Ironwood, Mich.:—I am sorry that I cannot take part in this discussion at this time, principally because I am personally interested in, and connected with, a lawsuit involving this subject, which will come before the courts of the State of Michigan in the very near future. I do wish to say, however, that this matter deserves the very serious consideration of this Institute and any other men who are equipped to handle it, as demonstrated by the fact that it has involved very serious consequences in the State of Michigan of great importance to the entire mining industry. It is obvious that a basis for a just method, or system, of mine-valuation for the purpose of taxation cannot be reached by jumping at conclusions. It should have the benefit of careful consideration by a body of men who are best fitted and equipped to handle it. It seems to me that this is the only way in which we can ever arrive at a fair and just method, an equitable method. The question is of vital importance to the entire industry of mining.

R. V. NORRIS, Wilkes-Barre, Pa.:—Would I be out of order if I were to say a word on the taxation of coal?

THE CHAIRMAN:—No, sir.

R. V. NORRIS:—In connection with this paper the present status of taxation of coal-lands in Pennsylvania may be interesting.

Land in Pennsylvania is assessed under the Acts of 1841 and 1842, under which the assessors are required to "Assess, rate and value every subject of taxation according to the actual value thereof, and at such rates and prices as the same would bring at a *bona fide* sale after due notice." Under that law the Pennsylvania authorities have undertaken to tax the coal in the ground, separately from the surface; the result has been an infinite tangle of lawsuits, and in the anthracite region the determination of the legal status of the assessment of 1907 is still before the courts.

For a number of years the valuation was made for assessment upon the basis of an acre of anthracite coal 1 ft. thick, which was used as a unit. This method was not objected to until the assessors in 1907 placed what the owners considered a preposterously high value upon a foot-acre of coal. On an appeal from this assessment the Pennsylvania Supreme Court, in the "Reading Appeal," says (Penna. Supreme Court Reports, No. 229, p. 465) ". . . that the foot-acre rule for ascertaining the value of coal lands of the appellant for the purpose of taxation is not the proper measure of their value . . ."

Attempts have been made to put into force a valuation based on the present value of the royalty which might be obtained on the property; in connection with this, on appeal, the Pennsylvania Supreme Court says (Penna. Supreme Court Reports, No. 229, p. 470):

"Its market value is its fair selling value for cash, not payable as royalty strung out through a long series of years but payable at the time or as soon thereafter as the value could be determined. Such a method does not make allowance in undeveloped territory for the length of time coal may lie in the ground unmined, undeveloped and unprofitable. It is impossible to reduce to a scientific basis and to mathematical precision the elements of value entering into the present selling price of a tract of coal land. The question is not what earning power coal lands may develop in the future but what they are actually worth in the market at present."

In spite of the fact that this matter has been referred to the Supreme Court at least three times, the local authorities are still unsatisfied, and have within the last three months pushed to trial a fourth appeal from the 1907 valuations. Under the Supreme Court decisions it has been necessary to show sale values of coal-land to reach a market value at public sale. This

has resulted in a mass of contradictory testimony as to sales, the prices received having ranged from a few hundreds of dollars per acre up to ten or twelve thousand, the smaller values having been usually for undeveloped coal-land and the large for going operations, in which many other considerations than value of coal alone were considered, or for tracts which were bought at large prices because of their interference with extensive operations. The result of all this litigation has been to convince many of us who have given the subject of mineral taxation serious attention that it is logically wrong to tax coal or any other mineral in the ground. Other resources, such as sand-banks, rock deposits, mineral springs, etc., are not assessed

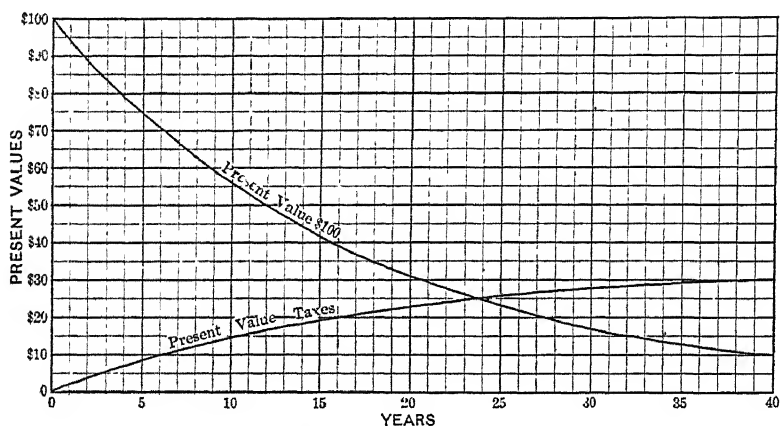


FIG. 2.—PRESENT VALUE OF \$100 FUTURE PAYMENT AT DIFFERENT PERIODS AND PRESENT VALUE OF TAXES ON SAME AT 20 MILLS FOR SAME PERIODS. CALCULATED AT 6 PER CENT.

by any such method, and the result of such taxation is to encourage the destruction of mineral deposits by the mining of the cheapest and most valuable beds, regardless of the influence of such mining on less desirable parts of the deposit, as, in coal mining, the removal of large beds regardless of overlying beds of poorer quality or less thickness which will be injured and often utterly destroyed by the mining of the large underlying beds. Such a course is opposed to all ideas of conservation, but with the present trend of taxation under the present laws is practically forced on the owners of mineral deposits.

The diagram, Fig. 2, shows graphically the present value of \$100 worth of mineral to be mined at a future date, and also

the present value of the taxes at the rate of 2 per cent., the present local rate, on such mineral, and indicates clearly the necessity from a financial standpoint of very prompt removal, even at the cost of large losses in ultimate output.

In my opinion, the proper method of taxation on all mineral operations would be, not a valuation on the mineral still remaining in the ground, but an assessment on the year's output at its actual sale price at the mine, on which amount taxes at the local rate could be assessed and collected without any uncertainty or litigation, and without regard to the amount of mineral tributary to the operation. This would further result in the payment of taxes in the greatest amount at the times and in the localities of greatest production, and consequent greatest population and greatest need of money.

E. W. PARKER, Washington, D. C.:—I might state in connection with this matter that the State of Florida undertook some years ago to tax the standing timber. At the present time they have no standing timber tax. Everybody is cutting their timber so it will not be taxed. My idea is not to figure a tax in such a way that coal which will not be mined within twenty-five years is estimated to have a present value of about \$5 an acre. The true solution is not to tax such mineral, but to encourage the miner to mine those seams of ore which should be mined, instead of tearing out the best seams and ruining the property to avoid the tax.

THE CHAIRMAN:—Mr. Norris's statement of the taxing of coal in the anthracite regions is very close, as I understand it, to the situation created by the author of the paper in his mine-valuation. At least the criticism of it is, that value is placed on undeveloped ore—and in some cases undiscovered ore—and each year a tax accumulating on such ore finally becomes so burdensome that the parties are discouraged from exploratory work, for the reason that as soon as they find ore they are taxed for it. I do not know whether that applies to Mr. Souder's case or not, but I am rather inclined to think it would be along that line.

Mr. Parker referred to the State of Florida. I would call attention, as a matter of interest, to the fact that the Legisla-

ture of the State of Pennsylvania has before it at present a proposed law placing auxiliary forest reserves under the protection of the State, so that any one, an individual or corporation, owning forest land will be encouraged to seize lands under the protection of the Forest Reservation Commission; to be recalled if they want to at any time. That is to say, the State is endeavoring to encourage forestry by not taxing the forest lands for the present, but keeping them under State supervision, so that, when the harvest is secured, the State will share in it.

E. E. WHITE, Ishpeming, Mich. (communication to the Secretary *):—Mr. Finlay's paper is at first glance reassuring to Lake Superior mining interests, so far as foreign competition is concerned, but his costs per unit showing that foreign competition is impossible will not stand analysis. I wish to go into Mr. Finlay's figures in detail and show why it is that other engineers may arrive at different conclusions. I will tabulate the costs per unit for the average Lake Superior and Cuban ores, as Mr. Finlay has done, using his figures where they are reasonable.

Cuban vs. Lake Superior Ores.

In figuring the cost of Cuban ore I will take as the cost of mining 27 cents per ton, as Mr. Finlay has done, although I believe this is excessive. Since the cost of initial plant is not included in this figure, and since there is practically no barren overburden to be removed, I believe this ore could be mined and loaded into cars by steam shovels for 10 cents per ton. I will also take the cost of nodulizing at Mr. Finlay's figure of \$1 per ton, which is probably reasonable.

Mr. Finlay's cost of water transportation per ton-mile is certainly excessive. The rate from Escanaba to Lake Erie for 1913 is 40 cents per ton, or 30 cents for actual transportation, deducting 10 cents for loading and unloading boats, which is certainly conservative for terminal costs at both ends. Since the distance is about 575 miles, the cost per ton-mile is only 0.0522 cent. The rate from Escanaba in 1912 was 5 cents less, and the rate per ton-mile only 0.0435 cent, but the boats may not have made any money in 1912, so I will use the higher

* Received Mar. 10, 1913.

price. This will be more favorable to Lake Superior ores with their shorter haul. Using this figure gives us 62.6 cents per ton for the 1,200 miles sea-haul from Cuba. This is reasonable, since tramp steamers, which are not built to carry iron-ore and which do not specialize in the traffic, are now contracting for this haul at 80 cents per ton. I believe the rate should logically be less than that on the Great Lakes, since on the sea there is the advantage of being able to use cheaper foreign-built boats, cheaper fuel, and cheaper labor. Foreign owners are satisfied with a smaller rate of interest on the investment than American owners, and there is also the very important point that these boats would operate the year round, whereas boats on the Great Lakes can only operate for about one-half the year.

Mr. Finlay's rate for land-haul is certainly excessive, since a rate is quoted from Baltimore to Pittsburg of \$1.25 per ton. The distance by the Pennsylvania railroad, the shortest route, is 328 miles, so that, allowing 15 cents per ton for terminal costs at both ends, which is low, the rate is 0.336 cent per ton-mile. I presume that the reason for this low rate is that there is considerable coal going East from Pittsburg, so that the cars are loaded both ways. For the purpose of argument I will assume that Lake Superior railroads could meet this rate in competition, though as a matter of fact they probably could not. Railroad men tell me that it costs about 0.55 to 0.60 cent per ton-mile in the Lake Superior district. It must be remembered that in this district there is, in general, freight going only one way and that the ore is all moved in about one-half the year, both of which add greatly to the cost. I will use a total land-haul of 234 miles for Lake Superior ore, instead of Mr. Finlay's figure of 228 miles, since the average haul to lake is 74 miles, not 68 miles, using his own preliminary figures.

As stated above, I will assume 10 cents per ton for loading and unloading boats, which is low, since this is the charge at Lake Erie for unloading only. I will take 18 cents per ton for terminal costs in Cuba and at the upper lakes. Railroad men tell me that the cost is from 18 to 20 cents per ton, due to the amount of switching at the mines and the cost of docks. For the land-haul to Pittsburg I will take 15 cents per ton terminal costs, as stated above.

Mr. Finlay slights the questions of royalty and taxes, which are very important. He evidently assumes that if it came to a question of close competition with Cuban ores the royalty and taxes on Lake Superior ore might be brought down to the Cuban level. This is entirely wrong. The royalty in the Lake Superior district is a fixed charge and cannot be reduced. Most of the mines are held under long term leases, and while theoretically the fee owners might reduce their royalty in competition with Cuba, this would not work out in practice. I will assume 30 cents per ton royalty in the Lake Superior district, which is low for the average ore, considering the high royalties on the Mesabi Range. I will take 5 cents per ton royalty for Cuban ore, which is certainly an outside figure, since enormous tonnages have been bought outright at less than 1 cent per ton in the ground.

In the same way, taxes in the Lake Superior district could not be reduced to any extent. I will take as the cost of taxes 10 cents per ton for the Lake Superior district, which I believe is conservative. If we considered only Michigan mines the cost per ton would be much higher, due to Mr. Finlay's excessive valuation for the Board of State Tax Commissioners, but the lower cost of taxes per ton of Mesabi ore brings down the average. Although there are no taxes now on Cuban ore, I will assume a cost of 5 cents per ton, since the mines would be taxed a certain amount when mining starts on a large scale.

Mr. Finlay takes no account of the differential on Mesabi ore, which should certainly be considered. Owing to its physical character, it is worth less than Old Range or Cuban ore of the same grade, both of which are coarser and more desirable for smelting. The differential on Bessemer ore is 25 cents per ton and on non-Bessemer ore 20 cents per ton. Since 70 per cent. of the average Lake Superior ore is Mesabi, this makes an average differential of 16 cents per ton which must be added to the Lake Superior cost to compare with the cost of Cuban ore.

Using these figures and assuming 52 per cent. iron natural for Lake Superior ore and 52.75 per cent. iron and nickel natural for the nodulized Cuban ore, as estimated by Mr. Finlay, we have the following costs per unit:

Cost of Average Lake Superior Ore at Pittsburg.

	Cents per Unit.
Mining, 87 cents per ton.....	1.67
Land-haul, $234 \div 0.336 = 78.6$ cents per ton. ...	1.51
Terminal costs at mines and upper lakes, 18 cents per ton.....	0.35
Terminal costs, Lake Erie and Pittsburg, 15 cents per ton.....	0.29
Lake-haul, $790 \times 0.0522 = 41.2$ cents per ton.....	0.79
Loading and unloading boats, 10 cents per ton.....	0.19
Royalty, 30 cents per ton.....	0.58
Taxes, 10 cents per ton..	0.19
	<hr/> 5.57
Premium on Bessemer ore, 48 cents per ton. Since the average Lake Superior ore is $\frac{1}{3}$ Bessemer, premium 16 cents per ton	0.31
	<hr/> 5.26
Average differential on Mesabi ore, $22\frac{1}{2}$ cents per ton. Since average ore is 70 per cent. Mesabi, differential 16 cents per ton..	0.31
	<hr/> 5.57
Total cost per unit to compare with Cuban ore.....	5.57

Cost of Cuban Ore at Pittsburg.

	Cents per Unit.
Mining, 27 cents per ton, 40 cents nodulized.....	0.76
Nodulizing, \$1 per ton.....	1.89
1,200 miles sea-haul at $0.0522 = 62.6$ cents per ton.....	1.19
25 miles land-haul in Cuba at $0.33\frac{1}{2} = 8$ cents per ton..	0.15
323 miles land-haul to Pittsburg at $0.336 = \$1.10$ per ton	2.09
Loading and unloading boats, 10 cents per ton.....	0.19
Terminal costs in Cuba, 18 cents per ton.....	0.34
Terminal costs in U. S., 15 cents per ton.....	0.28
Royalty, 5 cents per ton.....	0.095
Taxes, 5 cents per ton.....	0.095
	<hr/> 7.080
Premium for Bessemer quality, 48 cents per ton.	
Premium for phosphorus 0.025 per cent., 25 cents per ton.	
Total premium, 73 cents per ton.....	1.38
	<hr/>
Total cost per unit to compare with cost of average Lake Superior ore.....	5.70
Duty, 12 cents per ton.....	0.23
	<hr/> 5.93

These figures show that Cuban ore can compete on favorable terms with Lake Superior ore as far west as Pittsburg when the duty is removed. The difference, 0.13 cent per unit, is only 7 cents per ton, and might easily be saved in the cost of mining. If we should take into account the cost of initial plant, the result would be even more favorable to Cuban ore,

since a large part of the Lake Superior output is mined underground, with expensive shafts and equipment.

It is evident that the reason Mr. Finlay's costs for the transportation of Cuban ores are too high is that he has divided the flat Lake Superior rate for transportation and terminal costs by the distance. This is, of course, entirely wrong, since the terminal costs in the case of the longer haul of the Cuban ores are much less in proportion than on the shorter haul of Lake Superior ores. The terminal costs which I have assumed are conservative, and if we should take higher terminal costs, the result would be even more favorable to Cuban ores.

If Cuban ores do not compete with Lake Superior ores it will not be because of the comparative costs of mining and transportation, which are all that Mr. Finlay has considered, but will be because of undesirable qualities in the Cuban ore, which Mr. Finlay does not mention.

Internal Competition and Future Price of Lake Superior Ore.

Aside from the question of outside competition, there is the great question of competition from within, which is of greater importance than Mr. Finlay thinks. The production of Lake Superior iron-mines is at present, and will be for many years, limited by the consumption, not by the capacity of the mining plants, which could easily turn out a much greater production if the ore could be sold at a profit. This, of course, is more particularly true of non-Bessemer ores. It is a well-known fact that unless a Lake Superior iron-mine produces high-grade ore, it will have to hold its stock-piles over until the next season about one year in five, or sometimes even one year in three. It is impossible to sell more than a certain amount of any but high-grade ore, and it is common practice to hold over the year's production. Except in boom years, there is always a great quantity of ore in stock.

Within the next few years this condition of over-capacity will be even more marked, due at first, as now, to the rushing of mining operations on the Hill lease of the Steel Corporation to get out as much ore as possible before 1915, when the lease is to be given up, and after 1915 to the working of mines on the Hill lands independent of the Steel Corporation, which will then fall back on its other mines and still get out its

usual tonnage. It is a well-known fact that the Steel Corporation in many cases does not now work mines which it owns in fee. When the ore is needed it will open them up. Ore mined on the Hill lands after 1915 will therefore practically be in addition to the present output. There will also be large additions from the Cuyuna Range, which first started to ship in 1911.

For these reasons alone, neglecting foreign competition, the Lake Superior mines anticipate lower prices for iron-ore. The state of internal competition is apparent to those conversant with commercial conditions in the district, though perhaps not to Mr. Finlay. I believe Mr. Finlay's Fig. 1 is entirely misleading in as far as it is taken to show that the fluctuation in price is diminishing and is vibrating between an ascending minimum price and a horizontal level of effective foreign com-

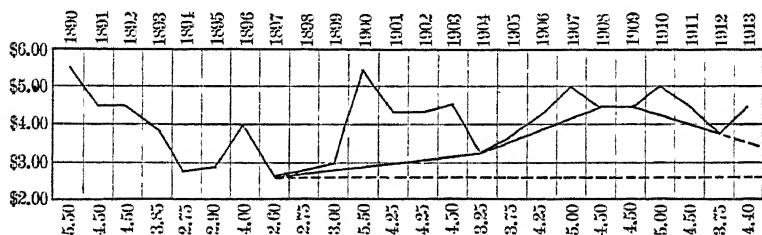


FIG. 3.—CHART OF IRON-ORE PRICES. MR. FINLAY'S FIG. 1 REVISED TO SHOW DECREASING MINIMUM SINCE 1909. PRICES ARE FOR OLD RANGE BESSEMER ORE.

petition. The line of ascending minimum prices appears to rise only because we have had no major panic and depression since 1893. It is not an accurate presentation of the facts in that it is not the true line of minimum prices, as stated. The line should rise from the low price of 1904 to the minimum of 1908–1909 and then drop to the low price of 1912, so that the present slope is downward, as shown in the accompanying diagram, Fig. 3. That this is the true course of prices is indicated by the fact that the price for 1913, expected to be a year of very good business, is even lower than that prevailing during the considerable depression of 1908. There are good reasons for this decline, as stated above.

I do not understand how Mr. Finlay can say that the price of Old Range Bessemer ore was fluctuating in 1912 above a

minimum of \$3.75 a ton, the price for that year. \$3.75 cannot be the present minimum, for 1912 was a year of very fair business. The record of shipments of Lake Superior iron-ore was broken and records were broken in other lines of business. Whenever we have a depression comparable to 1893, or even to 1908, the minimum price will surely sink to a lower level than at any time since 1904, and careful students of fundamental conditions expect such a depression. The present high level of wages and commodity prices, together with the labor unrest accompanying high prices, makes a reaction certain sooner or later. In this connection note that the price of iron-ore has not followed commodity prices to the new high levels, or to the level of effective foreign competition, as would be expected. In fact, the 1913 price of Old Range Bessemer ore has declined 12 per cent. from the price of 1907, whereas the average London Economist Index of commodity prices for 1912 shows an increase of 7.62 per cent. over the 1907 high record, and whereas Bradstreet's Index for December, 1912, was 9.5462, an increase of 4.67 per cent. over the high record of 9.12 in March, 1907. The percentage decline in price of the lower grades of Lake Superior ore has been even greater than that of Old Range Bessemer.

Adequate Return on Mining Investments.

From Mr. Finlay's remarks as to the greater variation in price of iron-ore than of gold it is evident that there is greater risk attending the mining of iron-ore in operations of commensurate magnitude where the ore bodies are blocked out well ahead. In view of this I do not understand how he can assume that 5 per cent. is an adequate return on capital invested in iron mining. To be sure, as Mr. Finlay says, a valuation is logical only on the assumption that the factors used in making it are correct, and the factors are as likely to err on the one side as on the other, provided the person making the valuation uses good judgment and has a knowledge of local as well as general conditions; but the very fact that the factors are only assumptions, and may vary one way or the other, constitutes the risk and necessitates more than an ordinary rate of interest. Mr. Finlay admits that an error of only 10 per cent. in estimating the average price of an iron-ore will often make far more differ-

ence in the valuation than any possible mistake as to the life of the mine. In the light of the foregoing discussion as to the expected future price of Lake Superior iron-ore, it may be seen how easily an error of even more than 10 per cent. may be made. Mr. Finlay estimates the average future price of Old Range Bessemer ore between \$4.50 and \$5 per ton, whereas I estimate an average price of about \$4 per ton. If my opinion should prove correct, Mr. Finlay's price would be from 12.5 to 25 per cent. too high. A difference of only 10 per cent. in price is the difference between profit and loss at many mines, and yet Mr. Finlay says that 5 per cent. is a sufficient return on the investment.

Many good industrial bonds can be bought to yield 5 per cent. and good railroad bonds to yield 4.5 per cent. where there is no question about getting interest regularly and money back at the end of a term of years. In the iron-mines one cannot be certain of getting interest regularly, or even of getting one's money back eventually, due to the variation in the price of ore, in wages and labor conditions, in taxation, and due to mine fires, cave-ins and other accidents, to say nothing of the ever-present possibility of over-estimating the amount of ore remaining in a mine, often based largely on the results of diamond drilling. Such tonnages are only estimates, and cannot be considered assured. Nobody will pay a price for a mine on which he estimates the profits will only pay 5 per cent. and return the money, when he can so easily buy bonds which will net the same amount and which he can put away and forget about except when time to clip the coupons. Bear in mind that the interest and money back on bonds are *assured*, whereas the profits on iron-mines are only *estimated*.

Mr. Finlay says that he does not wish to dismiss the question cavalierly, and then proceeds to do so. He does not appear to give any weight to the views of other eminent engineers on this point. G. A. Denny, an English mining engineer, speaking of mining in general, says: ⁵

"A normal mining risk, stated in terms of interest, may be taken at 10 per cent. per annum on the capital expended, plus a rate for the redemption of capital"

⁵ *Mexican Mining Journal*, vol. xi., No. 1, pp. 25 to 29 (July, 1910).

He further says:

"In the interest and redemption charges there should be shown a sum which is the interest at a given rate on the total capital expended plus the share capital, in cases in which profits have to be earned on a share capital, which, at parity, is in excess of the actual cash expenditure. The rate of interest will of course vary with the risk, but it should not be less than 10 per cent. per annum."

Speaking of mining in Mexico, he says:

"The terms of the bank's loan would call for:

"1. A higher than the ruling rate of interest for commercial business to cover the added risk, not less, therefore, in Mexico than 15 per cent. per annum.

"2. A repayment of the loan in annual installments over a period of years."

Since Mr. Denny has been particularly interested in the Rand in South Africa it is fair to assume that he includes that district in his general statement of mining risk. I think the Rand may be fairly compared with the Lake Superior district in the permanence of mining investments and the normal risk. In fact, the risk would be less on the Rand, since the value of gold does not fluctuate as does that of iron-ore, as mentioned by Mr. Finlay, and since the gold beds are of greater extent and more continuous than the pockets and lenses of iron-ore in most iron-mines, particularly those in Michigan.

W. L. Austin, speaking of mining in general, says:⁶

"It has been shown above that standard railroad shares may be purchased to-day in the open market at rates which give a return of 6 per cent. and over. Now a little consideration of the subject will convince any one that for various reasons a mining stock should pay a higher rate of interest than one issued by a trunk line of railway. What a proper rate would be, will, of course, have to be determined in each separate case; but in order to have a basis for estimating the approximate value of mining shares in general, let it be assumed that 10 per cent. is the minimum rate that should be expected from a mining investment. In this connection the remarks accredited to Mr. J. C. Stubbs—the railroad man—which appeared in the *Los Angeles Times* of the 18th inst., are pertinent. Mr. Stubbs is reported to have said:

"What manufacturer or merchant would accept the risks of business for 6 per cent. on the money invested?"

"And it is claimed for the better class of mining enterprises that they approximate manufacturing industries in their essential features?"

"Let the case of a mining investment be examined more closely where the investor is to receive 10 per cent. for the use of his money, and where the principal is to be recovered upon the exhaustion of the mine. Many business men would not consider this return sufficient, in view of the risks inseparable from mining, but when it is explained how the results hereinafter given are arrived at, other values can be substituted in the formula to meet individual views."

⁶ *Mines and Methods*, vol. ii., No. 3, p. 56 (Nov., 1910).

John Hays Hammond, speaking of mining in general, says:⁷

"In many mines persistency of the ore deposits and, therefore, the reliability of the mines as dividend payers, justifies the investment upon a basis in some instances as low as 8 per cent. dividends, to which, of course, must be added a certain percentage to provide for the amortization of the capital. Generally speaking, however, investments in mining securities are not to be regarded as attractive unless they return from 10 per cent. to 15 per cent. in dividends, in addition to the profits to be set aside for amortization."

I have failed to find anywhere any written statement by any other well-known mining engineer that 5 per cent. is a reasonable return on a mining investment. I believe the assumption is indefensible.

Example of Mr. Finlay's Method of Mine-Valuation.

Mr. Finlay's method of valuing mines is reasonable when used with proper factors, but the factors which he assumes and certain errors which he makes lead to exaggerated valuations. I can best discuss these factors and errors by taking the concrete case of a mine valued by Mr. Finlay, and will take as an example a certain mine valued in connection with the valuation of the mines of Michigan for the Board of State Tax Commissioners. Mr. Finlay's valuation of this mine was certainly wrong, and imposed excessive and unjust taxes. The taxes in 1911 for this mine were 22.5 cents per ton on 350,926 tons, by far the largest production in its history, and a good-sized output for any Michigan iron-mine. In 1912 270,232 tons were mined, still a large output, yet even after a reduction in value by the assessors for ore mined out since Mr. Finlay's valuation, the taxes came to 27.8 cents per ton.

Mr. Finlay estimated the cost of producing ore at the mine in question at \$2.73 per ton, delivered at lower lake ports, but this only included 21 cents for general expense and taxes, as against the 22.5 cents to 27.8 cents for taxes alone which his increased valuation imposed. As a matter of fact, Mr. Finlay, in making this valuation—and I believe in all his valuations of Michigan mines—made no allowance for the increased cost per ton for taxes due to the increased valuation. The average general expense reported to him for the past five years at this mine was 12.1 cents per ton and the average taxes 9.1

⁷ *Engineering and Mining Journal*, vol. lxxxix, No. 1, p. 10 (Jan. 1, 1910).

cents—total, 21.2 cents per ton, which agrees with his estimated 21 cents for the two. Considering for the present the rest of his estimate of cost and his average selling price correct, these increased taxes alone would reduce his estimated profit from \$1.03 to 87 cents per ton and his valuation from \$7,000,000 to \$6,100,000. Since the taxes paid on this mine since Mr. Finlay's valuation are about \$75,000, this error alone costs the operating company \$10,000 per year.

But the price of ore was so low in 1912, following Mr. Finlay's valuation, that, in place of his estimated selling price of \$3.81 (note the discrepancy between his estimated cost, selling price, and profit), the ore mined was only worth an average of \$3.19 per ton at Lake Erie ports, reducing the profit—still using Mr. Finlay's estimated cost of \$2.73 per ton plus increased taxes—to 30 cents per ton. Does it not seem that 27.8 cents per ton for taxes was confiscatory when the profit was only 30 cents per ton? As a matter of fact, this "profit" was only just sufficient to meet the royalty. I realize that the price for ore in 1912 was low, but this does not alter the fact that an unjust proportion of profits was paid in taxes. Mr. Finlay may say that this was because the production was only 270,232 tons in 1912, against his estimated 500,000 tons. I can only say that 270,232 tons is a big output from any underground mine in Michigan with a big water problem, with a heavy sand capping, and disturbed by frequent sand runs. Since the lease ran 43 years from January 1, 1911, when Mr. Finlay estimated the mine to have 13,635,000 tons of ore, it is only necessary to take out 317,000 tons per year, on the average, against Mr. Finlay's estimated 500,000 tons, and because of difficult mining conditions and difficulty in selling such a large quantity of ore, mostly non-Bessemer, it is very doubtful if Mr. Finlay's estimated tonnage per year can ever be reached.

But this question of the cost per ton of the increased taxes was merely an error, or oversight, and is not nearly so important as the assumptions already discussed which Mr. Finlay makes in his valuations. He arrives at his cash value by multiplying his estimated production per year by his estimated profit per ton and then by the present value of a \$1 per year dividend during the life of the mine, calculated to return 5 per cent. on the investment and replace capital when the mine is

exhausted by the investment of an annual sum at 4 per cent. I believe I have shown that it is reasonable to expect 10 per cent. return on capital invested in iron-mines. Let us see what difference it would make in the valuation of this mine by Mr. Finlay's method, assuming a 10 per cent. return instead of 5 per cent. Mr. Finlay figured the life of the mine as 27 years and the production as 500,000 tons per year. I will use this assumed life and production, although, as stated above, the average annual production will probably not exceed 317,000 tons. The present value of \$1 per year for 27 years to return 10 per cent. interest and replace the capital in 27 years by reinvestment of an annual sum at 4 per cent. is \$8.25, against Mr. Finlay's \$14.04 to return 5 per cent. interest and replace the capital in the same way. Using this lower factor and the decreased profit per ton due to increased taxes, a change which must be admitted to be correct, but using all other factors as assumed by Mr. Finlay, his value of \$7,000,000 is at once reduced to $500,000 \times \$0.87 \times 8.25 = \$3,600,000$.

In passing, I may say that I do not agree that the sinking fund to replace capital may be assumed to draw 4 per cent. interest. There is also a difference of opinion among mining engineers as to the number of years in which capital should be returned, some contending that it should be returned in not over ten years, regardless of the life of the mine indicated by ore reserves. G. A. Denny, writing in the *Mining Journal* (London) on The Commercial Aspect of Rand "Profits,"⁸ says:

"Mining, even of the safest character, is hedged around with many and various uncertainties, and even if the value of the ore is regular, and the deposit unaffected by dikes and faults—which is rarely experienced—extraneous conditions and events, such as wars, strikes, etc., may create heavy losses both directly, because of destruction or damage to property, or indirectly, because of interest losses.

"To insure himself against these possibilities, the mining investor should satisfy himself that the 'life' of the mine, as computed by the responsible technical advisers of a given company has been fully conveyed to shareholders (because it sometimes is not), and he should calculate his capital redemption payments accordingly; but in no case should he permit the capital replacement fund to accrue over a longer period than 10 years. He must also invest the redemption fund in the most secure, and therefore the lowest, interest-paying securities, and in any case never calculate to replace capital at a higher rate than 3 per cent."

⁸ Abstracted in *Engineering and Mining Journal*, vol. lxxxv., No. 9, p. 446 (Feb. 29, 1908).

Suppose we assume that capital invested in this mine should be returned in ten years, as stated by Mr. Denny, at the same time returning 10 per cent. on the capital. In view of the possible competition of Cuban ores in a few years, I believe this is reasonable. The present value of a \$1 dividend for ten years would be only \$5.45, assuming the sinking fund invested at 4 per cent.; or \$5.34, assuming the sinking fund invested at 3 per cent., as recommended by Mr. Denny. The present value of the mine, still assuming Mr. Finlay's estimated production of 500,000 tons per year and his average selling price, would then be $500,000 \times \$0.87 \times 5.34 = \$2,320,000$.

Now we come to the question of the future value of ore in the iron-mines of Lake Superior. Mr. Finlay says in his report to the Board of State Tax Commissioners of Michigan, speaking of iron-ore :

"The expected future price of ore has been fixed for the purposes of this report on a calculated value for all grades based on the average quotations for standard ores for the last seven years. . . . The question arises whether the average price for the last seven years is a fair one for the future. As in the case of copper, the prices ruling for the season of 1911 are lower than the calculated average used. The considerations that impel me to use this average are precisely the same as those used in the case of copper, there being the same reasons to expect a continuance of the demand in the one case as in the other."

I believe seven years is too short a time, and the seven years taken by Mr. Finlay happen to be the best seven years in the iron-ore business since the seven years beginning in 1887, when standard Bessemer ore was worth \$6 per ton. This was at the end of the early period of high prices in the infancy of the iron industry in this country. If Mr. Finlay in determining the expected price for copper had taken the last seven years, ending either in 1910 or 1911, as he did for iron-ore, his average price for lake copper would have been 15.5 cents per lb.,⁹ instead of 14 cents per lb. This would have increased the value of Mr. Finlay's fourteen profitable copper-mines 38 per cent., since the average cost of production of these mines was 10.1 cents per lb. As a matter of fact, Mr. Finlay took a longer period to determine the average price of copper. He states that the average price for the 33 years ending 1910 was 13.5

⁹ *Stevens's Copper Handbook*, vol. x., p. 1879 (1910-11), or *Mineral Industry*, vol. xix., p. 157 (1910).

cents; also for the 20 years ending 1909. He states that for the 15 years ending 1910 the average quotational price was 14.3 cents, so that his average of 14 cents must have been taken over a period of between 15 and 20 years. As a matter of fact, statistics show that the average has not been as low as 14 cents for any period of years ending 1910 less than 18 years,¹⁰ except for the three years 1908 to 1910, inclusive, which is manifestly too short a period upon which to base an average price. Since such a long period was taken to determine the true average price of copper, why not of iron also? It is true that the last seven years included high copper prices, but the same is true of iron-ore. This period does not include a major depression, but only the "undigested securities panic" of 1907-1908, which was of short duration and had very little effect upon commodity prices or upon the price of iron-ore.

If Mr. Finlay had taken 18 years instead of seven for iron-ore his average price for standard Old Range non-Bessemer ore would have been \$3.16 instead of \$3.77.¹¹ Mr. Finlay thinks that the course of iron prices is upward, but this is only his opinion, and I have shown that the reverse is just as probable. It looks as though Mr. Finlay had taken a shorter period for the iron-mines than for the copper-mines to get a higher price to fit his theory. As a matter of fact, the price of standard non-Bessemer ore was \$3.05 in 1912, 72 cents lower than Mr. Finlay's estimate of \$3.77, and is \$3.60 for 1913. Both are years of fair general business, with the record for iron-ore shipments from Lake Superior broken in 1912, and probably to be broken again in 1913, even at the low prices prevailing. 1913 is expected to be a boom year and yet the price for iron-ore is 17 cents lower than Mr. Finlay's estimate of the average price. Note that if Mr. Finlay had taken the average price for 18 years it would have been \$3.16, which compares very closely with the actual average for 1912 and 1913 of \$3.325 per ton.

This personal opinion of Mr. Finlay's made a big difference to the iron-mines of Michigan. Taking the same mine for an example, using an 18-year period, the same as the copper-mines, Mr. Finlay's estimated selling price of \$3.81 for this ore would be reduced to \$3.20, and the profit, allowing for increased

¹⁰ *Idem.*

¹¹ *Hurd's Iron Ore Manual* (1911).

taxes, would be reduced from \$1.03 to 31 cents per ton. Figuring the value again by Mr. Finlay's method, still assuming his life and yearly output but assuming 10 per cent. return on the investment, gives a cash value for the mine of $500,000 \times \$0.31 \times 8.25 = \$1,280,000$, against Mr. Finlay's \$7,000,000. This shows the extent to which Mr. Finlay's personal opinions as to adequate return on a mining investment and the future price of iron-ore affected the value of one mine in Michigan.

Another thing in Mr. Finlay's valuations which makes a big difference is the tonnage per year assumed. As stated above, 500,000 tons a year for the mine in question was altogether too high. Take this mine again for an example and figure the valuation by Mr. Finlay's method, using the probable yearly production of 317,000 tons per year for 43 years. Assuming a profit of 31 cents per ton, which was figured above by using the average selling price for 18 years and Mr. Finlay's estimated cost plus increased taxes, and assuming a 10 per cent. return on the investment, the value of the mine at the time of Mr. Finlay's estimate would only have been $317,000 \times \$0.31 \times 9.17 = \$900,000$. If we should decide that the capital should be returned in ten years, the sinking fund being invested at 3 per cent., as claimed by Mr. Denny, the value would only be $317,000 \times \$0.31 \times 5.34 = \$525,000$. Mr. Finlay may say that although 500,000 tons were not mined in 1912, it could have been done. Of course, an operating company realizes as well as Mr. Finlay does that the quicker it can get its ore out of the ground, provided it can do it at a profit, the more money it is making for its stockholders. The natural inference is that the operating company could not mine and sell at a profit 500,000 tons of ore from this mine in 1912. It has not approached this figure at any time in the history of the mine, mainly because of the difficulty in selling such a large quantity of non-Bessemer ore, due to the over-capacity in the Lake Superior district discussed above.

My figures show what an enormous difference in valuation is made by different assumptions in using Mr. Finlay's method. I believe I have shown that the assumptions made by Mr. Finlay in valuing Michigan mines were wholly wrong and greatly exaggerated the values of the mines. The taxation

based upon these values is unjust, and will continue to place a heavy burden upon Michigan mines till they are corrected. A word from Mr. Finlay that he feels that his assumptions may have tended to exaggerate the values of the mines would undoubtedly help a great deal in repairing the injustice that has been and is being done.

Summary.

To sum up, I would say that Mr. Finlay's estimate of the cost of laying down Cuban ore in Pittsburg is altogether too high, principally because he did not know the rates for transportation. Unless Cuban ores prove to be undesirable on account of the nickel and chromium, or for other reasons, they will be absolutely competitive with Lake Superior ores as far west as Pittsburg. As a matter of fact, I understand that in certain cases a premium has been paid for steel from Cuban ore on account of the nickel and chromium.

I believe Mr. Finlay's deductions as to the future price of Lake Superior iron-ore and his assumptions as to adequate return on capital are entirely wrong. His method of valuation is reasonable when used with proper factors, but the factors which he assumes in applying this method are not justified. The factors necessary to apply his method to any mine are five:

1. The average cost per ton.
2. The ore reserves.
3. The production per year.
4. The average selling price.
5. The present value of a \$1 per year dividend, depending upon the rate of interest on the investment, the number of years in which capital should be returned, and the rate of interest at which the sinking fund may be invested.

Of these five factors, as applied to Lake Superior iron-mines, the first two may be determined with fair accuracy, although in many cases the ore reserves are only estimates based on drilling, and although the average cost may only be determined by past experience, and may be different in the future, due to varying cost of labor, increasing cost and poorer quality of timber, and the possibility of even greater unjust taxation. In

this connection note the present agitation for a tonnage tax in Michigan and Minnesota, and the increased cost of labor since Mr. Finlay's valuation, due to the raise in wages Feb. 1, 1913, and to the Workmen's Compensation Law passed in Michigan in 1912. In general, however, we may consider these first two factors as known facts, and I do not object to the figures which Mr. Finlay used, except that he made no allowance for the cost of increased taxes, although he introduced the element of personal opinion even into these by using a lower cost per ton in most cases than was reported to him.

The next factor, the production per year, in the case of operating mines, is a known fact, but Mr. Finlay made it instead only a personal opinion by assuming greater production in many cases than has been produced in the past. It may be his opinion that a larger tonnage per year could be produced and sold at a profit, but there is certainly room for a difference of opinion, considering the foregoing remarks as to the difficulty of selling ore. It is my opinion that the actual average of shipments for the last five years should have been used. In the case of the mine which I have used as an example this was 299,500 tons per year, against Mr. Finlay's estimated 500,000 tons.

The last two factors, the average selling price and the present value of a \$1 per year dividend, I have shown to be purely matters of personal opinion. Mr. Finlay's valuations are therefore based on two-fifths known facts and estimates and three-fifths personal opinion. Since the valuations are three-fifths personal opinion, the last factor should certainly be based upon 10 per cent. return on the investment.

I believe Mr. Finlay's method of valuation may be successfully used, but the five factors for operating iron-mines in Michigan should be determined as follows :

1. The average cost of production at lower lake ports for five years, plus or minus the difference in cost per ton of taxes due to such revaluation.
2. The estimated ore reserves: ore based on diamond drilling to be estimated very conservatively.
3. The average production per year for the last five years, if the mine has been equipped to produce actively for that

length of time ; otherwise, for the number of years during which it has been so equipped.

4. The average selling price at lower lake ports for 18 years.
5. The present value of a \$1 per year dividend based upon a 10 per cent. return on the investment and capital returned in ten years of operation by investment of an annual sum at 3 per cent. This would mean 12 to 15 years from the beginning of development before capital would be replaced.

J. K. FINLAY, New York, N. Y. (communication to the Secretary *) :—My answer to Mr. White's criticisms may be put in a very few words. It may be the fault of my own wording of the article, but Mr. White has completely misunderstood the following points which I thought would be more or less self-evident to any one undertaking a serious discussion of the subject :

1. If a rate of \$1.25 per ton has been made from Baltimore to Pittsburg, is it not to attract some business which has not yet materialized ; to get some tonnage for empty cars ? What comparison will there be in the rate per ton-mile when Cuba undertakes to market 45,000,000 tons of ore each year in the United States ? When that time comes and Cuban ores are hauled at half the rate charged to Lake ores, will the Lake Superior people object or will they not ? Common sense will supply the answer.

2. Taxes and royalties as a part of the cost :

Will Cuban ores pay any taxes or any royalties ? Is it not self-evident that they must ? Is it not merely begging the question to assume that these items will be lower in Cuba than in Lake Superior ?

As a matter of fact, no property which is not profitable can pay either royalties or taxes, therefore these items are part of the profits. If some properties in Lake Superior pay higher royalties and taxes than they can afford that is unfortunate for them ; but it has absolutely nothing to do with a broad discussion of the fundamentals of valuation.

3. It is equally absurd to base an argument on changes in wages.

* Received May 5, 1913.

4. Mr. White thinks that a Lake Superior iron property—a going concern—should net 10 per cent. interest plus all amortization charges on its purchase price. This is an echo of other people's arguments to the same effect. It is a good argument for the tax dodger, but the prospective purchaser will find it a poor one. The prospective purchaser would find it particularly poor if he should try to use it coupled with Mr. White's opinion that I have estimated the value of Michigan ores some 50 cents a ton too high. According to this the profits would be only about 55 cents a ton, including royalties. On an average output of 11,000,000 tons a year the profits would be only \$6,000,000. On a 20-year life these mines would be valued by Mr. White at about \$45,000,000. On a 30-year life he would raise the price to \$51,000,000. On the average life at which I valued them, 18 years, he would obtain a value of \$42,000,000. The reader should recollect that these valuations include the royalties, and therefore the fee, of all the iron-mines of Michigan with their equipment and business.

In my judgment, these properties cannot be bought for three times any of the above valuations.

R. B. BRINSMADE, Puebla, Puebla, Mexico (communication to the Secretary*):—As I have previously considered the general equations for mine-valuation,¹ I will here discuss only that part of Mr. Finlay's paper criticized by E. E. White, along with some of the general principles of mining taxation, under the following six topics:

1. Rate of interest earned by sinking-fund.
2. Rate of interest earned by investment.
3. Place of royalty item in valuation.
4. Place of taxation item in valuation.
5. Calculation of future ore-prices.
6. Valuation of undeveloped mining land.

1. *Rate of Interest Earned by Sinking-Fund*—Mr. Denny's quoted opinion of a 3 per cent. rate is evidently based on English conditions before the Boer war. The great destruction of liquid capital by the great wars and catastrophes since 1897, along with the flood of gold and consequent rise in the average

* Received Aug. 2, 1913.

¹ Calculation of Mine-Values, *Trans.*, xxxix, 243 (1908).

index of commodity prices, has undoubtedly increased the rate of interest at least one-half per cent. In the United States sound savings banks now allow 4 per cent. on deposits, while safe bonds are quoted at prices which yield even 5 per cent.

2. *Rate of Interest Earned by Investment.*—Gold-mines with uncertain and scanty ore reserves, the objects of Mr. Denny's quoted remarks, are quite different from iron, copper, or coal mines with definite mineral reserves blocked out ahead by drilling or driving. While 10 per cent. is often an inadequate yield from gold-mines, the latter class of mines are often as safe as city real estate and resemble it as an investment in more ways than one.

City real estate is also speculative, for a shifting of the favorite locality for wholesale or retail trade will mean a corresponding change in the value of any included building lot. Yet 5 per cent. is now the common rate used by agents in figuring the value of a city business lot of a given rental yield. The investing public is evidently satisfied with a net yield around 5 per cent. from Michigan mines, for during many years the stock of the Calumet & Hecla Co. has been often quoted at prices to yield only 7 per cent., and this includes the necessary sinking-fund annuity.

3. *Place of Royalty Item in Valuation.*—As a mining royalty is equivalent economically to the ground rent of a building lot, Mr. White's idea of adding the royalty to the mining cost, thus subtracting it from the profits figure used to reckon valuation, would be the same as exempting the land-value of the mine from taxation.

City property is valued for taxation by capitalizing the total rent, which includes both ground rent and the hire of the buildings and improvements. The only difference for mine-valuation then should be the subtraction of an annuity item (to atone for the exhaustion of the mineral land-value) from the net operating profit in order to get the sum corresponding to the total rent of city real estate, which is capitalized for the assessment valuation.

In my recent discussion² of Dr. Raymond's article I have shown by means of the trinitarian diagram of economics that

² Our National Resources and Our Federal Government, *Trans.*, xliv., 633 (1912). .

rent is a residual factor, obtained by subtracting wages and interest from the total output, and therefore cannot under ordinary competitive conditions enter into the fixing of prices. Mr. White, by adding the royalty, or land-rent, item to the mining cost is thus cherishing the illusion that royalty is a factor *necessarily* affecting the scale of prices of his competitors, instead of being merely the deduction which some fee owners are able to make from the net operating profit.

4. *Place of Taxation Item in Valuation.*—Again referring to practice in city real estate, we find that the *current* taxes are deducted from the gross total rent in order to get the net total rent used as the basis of valuation. Mr. White's criticism here is evidently valid, but, owing to the varying rates of ore production, I believe his method, of adding the tax item to the mining cost, is less practical than to figure the rate of taxation in the equation for present worth of \$1, as suggested below.

Let V = value or present worth of a \$1 dividend to be assessed by taxation.

a = annuity to be paid to sinking-fund.

r = rate of interest earned on sinking-fund.

R = rate of interest earned on investment.

t = current rate of taxation.

n = number of years dividend is to be earned.

Then by suggested system $\$1 = (R + t) V + a$ (A)

And from algebra $a = \frac{Vr}{(1 + r)^n - 1}$ (B)

Substitute in (A) the value of a in (B) and

$$1 = (R + t) V + \frac{Vr}{(1 + r)^n - 1} \quad (C)$$

Solving (C) for V and

$$V = \frac{1}{R + t + \frac{r}{(1 + r)^n - 1}} \quad (D)$$

In equation (D) the system of Mr. Finlay gives us R and r , while t can be assumed approximately for the preliminary calculation. When the total preliminary valuation of Michigan has been made the current tax rate can be calculated directly from the necessary budget. With this exact figure for t it is

easy to get the correct figure for V by reckoning equation (D) a second time.

5. *Calculation of Future Ore-Prices.*—Mr. White seems to me unnecessarily worried over Mr. Finlay's assumed future base of \$4.50 per ton for iron-ore. As Mr. Finlay hints on p. 75 of his original report,³ there is no reason why mining property should not be reassessed annually by his system, as is everywhere the American custom for ordinary real estate. Any erroneous assumption of the future price of ore or cost of mining will thus only affect taxation for the current year. Another advantage of annual re-assessments would be that both the ore sold and the ore discovered in the previous year could be reckoned in the revaluation.

6. *Valuation of Undeveloped Mining Land.*—On p. 12 of his original report, Mr. Finlay says of undeveloped mining land that "no definite appraisal can be made." Here I differ, for a definite appraisal must be made if speculation and the hamstringing of development are to be discouraged. The most practical method of appraisal seems to be "self-assessment," under which the land-owner sets his own value, with the safeguard of the State's right to purchase during the year at the assessed value. A similar plan has been suggested by E. B. Kirby.⁴

Summary.

The sustaining of Mr. Finlay's general scheme of mine-valuation by the Michigan courts marks such an advance in the equitable taxation of mines as did the Ford franchise-tax law of New York in the taxation of public utilities. The fact that State assessing officials have usually been either incompetent or corrupt accounts for the previous under-assessment of mines and public utilities as compared with farms and homes.

The present trend in such progressive and democratic countries as Great Britain, Germany, and the self-governing British colonies is toward placing the entire cost of government on land-value owners, because they are the only class who financially benefit from government activity, since they absorb the

³ *Appraisal of Michigan Mining Properties*, by State Board of Taxation (1911).

⁴ *The Principles of Mine Taxation*, *Engineering and Mining Journal*, vol. xcii, No. 18, p. 853 (Oct. 28, 1911).

individually unearned increment from natural resources. The reform is best effected practically by gradually exempting true capital (buildings, improvements, etc.) from taxation and correspondingly increasing the tax rate on land values. This new system is explained generally by T. G. Shearman in *Natural Taxation* and for mining by an article⁵ I published in 1909.

The increase in the valuation of Michigan mines may be unfortunate for any operators who, disregarding the advancing democratization of taxation, were unwise enough to take long leases on a fixed royalty with the proviso that the lessee should pay all taxes, but even these may console themselves with the thought that Michigan improvements are not yet exempt from taxation.

By Mr. Finlay's system, the mine whose ore is produced at a loss is valued at nothing, even though its surface equipment may have cost a large sum, as in the case of many of the Michigan copper-mines. This system is indeed beneficial to producers, for improved mines are thus exempt from taxation until they become profitable; but is it not most discordant with the accepted theory of the general property tax? On the same principle, should not unprofitable buildings or machinery in other industries, such as manufacturing or commerce, be also called worthless and exempt from taxation? What a fortunate occurrence, if the controversy over Mr. Finlay's valuations results in a public exposure of the absurdity of the general property-tax system for a community of democratic producers!

⁵ *Natural Taxation of Timber and Mining Land*, *Mining World*, vol. xxxi, No. 21, p. 1023 (Nov. 20, 1909).

Blast-Furnace Slag-Analyses for 24 Hours.

BY F. L. GRAMMER, LEESBURG, VA.

(New York Meeting, February, 1913)

THE analyses given in Table I. were made several years ago at my request at a plant using Lake ores. They are of two furnaces, one making basic, the other Bessemer pig—they gave 6 casts each in 24 hr., also two flushes of cinder between each cast. They are offered simply for the sake of reference as, so far as I know, it is unusual to find so frequent full analyses made and none are on record.

The variations in manganese-content in the first and second flushes on the basic furnace are interesting, if no more than a freak—the first flush being invariably higher in manganese.

The iron-content of the two slags from an average of 12 flushes shows:

	Basic Per Cent.	Bessemer. Per Cent.
Total iron.....	2.85	1.70
Combined iron.....	0.72	0.52
Free iron.....	2.13	1.18

The greater iron-loss in the basic slag, while known, has never heretofore been so clearly set forth. This may modify the arrangement for handling slag; heretofore usually determined by proximity of the dumping-heap. If a nearby heap is used on contiguous territory, in the past, tilting molten-slag pots were used. If the Pittsburg flood-records for 30 years are a fair criterion the government regulations were inadequate, and molten slag was dumped in such a manner as to throttle the Monongahela at the time of the spring thaws.

At Pittsburg the slag, where it is carried to a distance, is frequently granulated in pits and then removed in cars to which it is conveyed by clam-shell buckets. This practice is also quite customary where the slag is intended for the manufacture of slag cement. The production of 500 tons of cast-iron daily means about 300 tons of slag and at 2.13 per cent. equals 6.4 tons

of free iron lost from the basic furnace daily, and about 3.5 tons from the Bessemer furnace, if these furnaces are representative.

Formerly considerable metal was brought in from the cinder dump, particularly from pots filled at the casting-time from drainings of the skimmer and the trough.

It seems to me that a nearer complete recovery of free iron should be possible in slag granulation with a magnet than with molten slag and dumps. Additional arguments may be found for granulation at basic furnaces.

The analyses of the casts show that the furnace was in a healthy condition as regards regularity. I do not presume much "buckshot" was being made. It is, however, impossible for me to say, as it was not for this purpose these analyses were made. These data were brought out with the intention of publishing them without comment. The relation between the percentages of iron caused a digression.

TABLE I.—Comparative Record of a Day's Run of Furnaces on Basic and Bessemer Irons.

Basic Furnace.				Bessemer Furnace.			
Tonnage on date of analyses, tons.				Tonnage on date of analyses, tons.			
Average for month, daily.				Average for month, daily.			
Mesabi, per cent.				Mesabi, per cent.			
Yield, per cent.				Yield, per cent.			
Coke per ton.				Coke per ton.			
Stone per ton.				Stone per ton.			
Cu. ft. air per minute.				Cu. ft. air per minute.			
Blast temperature, degrees F.				Blast temperature, degrees F.			
Blast pressure, lb. per sq. in.				Blast pressure, lb. per sq. in.			
512				520			
475				487			
81				69			
51.9				51.0			
2,234				2,289			
1,002				1,271			
35,000				37,500			
19.5 — 21.5				17 — 20.5			
Analyses of Iron.				Analyses of Iron.			
Comb. Fe.	MnO.	SiO ₂ .	Al ₂ O ₃ .	CaO.	MgO.	S.	Total Fe.
0.75	1.60	35.60	16.50	40.00	3.90	1.01	1.87
0.69	0.90	36.00	16.62	41.30	3.38	1.20	2.98
0.64	1.18	35.30	14.23	43.70	3.68	1.09	3.06
0.22	0.92	36.40	15.89	41.50	3.60	1.43	1.75
0.56	1.40	35.20	15.40	42.80	3.18	1.08	2.23
0.60	1.04	34.40	16.15	43.80	2.80	1.17	3.12
0.59	1.50	35.20	13.86	44.20	3.74	1.05	3.64
0.56	0.96	35.60	15.60	42.10	3.46	1.23	2.81
1.85	1.90	37.60	14.36	38.60	3.96	0.60	4.26
1.30	1.28	36.80	15.25	40.00	3.90	1.03	3.71
0.27	0.48	34.50	14.80	45.10	3.60	1.44	3.04
0.63	0.38	31.00	15.30	48.00	3.32	1.39	1.71

Full Slag-Analyses of all Flashes: Limestone analysis: SiO₂, 4.07; Al₂O₃ and Fe₂O₃, 1.50; lime, 47.87; magnesia, 3.42; phosphorus, 0.007.

The Microstructure of Sintered Iron-Bearing Materials.

BY B. G. KLUGH, BIRDSBORO, PA.

(New York Meeting, February, 1912)

THE present paper represents a preliminary investigation of the ultimate structure of iron-bearing materials which have been subjected to heat treatment for the purpose of agglomerating the fines, or for chemical changes, or for both.

Primarily, the object of this inquiry was to determine the relation of the microstructure of each of these materials to its specific susceptibility to the action in the blast-furnace. It is hoped that this small contribution will serve as a nucleus to a line of research that will throw some light upon the debated questions involved in blast-roasting.

Thus far the investigation has been conducted along lines of analogy of known principles, rather than through the application of any well-defined system of microscopic petrography. This is obviously due to the fact that we are dealing with materials which are the artificial products of a new art, rather than with the natural materials from which the essence of the pure science has been evolved.

In the limited time allowed for this research, it has been conducted upon those materials the chemical content of which allows of their use commercially in the blast-furnace.

The following microphotographs are arranged in the order of the degree of complete fusion to which the mass in question has been subjected, in effectively binding the particles together. A very notable fact here clearly demonstrated is that the degree of fusion to which the material has been subjected is inversely proportional to the reducibility of the product. The characteristic of the material under consideration to which reducibility is almost universally attributed can be progressively traced in the following arrangement, through the identity of the obvious microscopic constituent.

Furthermore, the degree of fusion to which a body has been subjected is recorded in the presence of resultant products of fusion in that body. Such readjustment is generally apparent

only in microscopic proportions. When two reacting bodies are brought together, the amount of the product of chemical union will essentially be in proportion to the time they are held in contact at a temperature adequate for their combination. Thus it follows that the greater the amount of fuel expended in agglomerating fine particles, the greater will be the amount of glass-like and impervious material found in the product. This predominating glass will naturally tend to form a glaze over all surfaces to which the gases are subjected, rendering the material more difficult to reduce. Therefore it is plainly seen in the analogous reasoning, and further demonstrated by concrete example, that the agglomerating process requiring for effective binding the least fuel is not only the most economical in its own operation, but produces the ideal material for the blast-furnace.

In all of the accompanying microphotographs the uniform magnification of 40 diameters is adhered to, which allows a fair comparison of pores, cells, crystals, masses and other characteristics.

Fig. 1 is a microphotograph of heating cinder magnified 40 diameters. In heating-furnace cinder, which is produced by the iron oxide combining with the chemical equivalent of silica in the furnace bottom to form the fusible, glass-like iron silicate, we recognize in the almost pure chemical compound the most impervious and least reducible of all commercial iron-bearing materials charged into the blast-furnace.

In Fig. 1 the most predominant constituent is the crystalline iron silicate, which is obviously the product of complete fusion, and slow cooling from the liquid state, with the free segregation of the eutectic. This is the characteristic of the heat-treated oxide mixtures which renders them least pervious to gases. Unfortunately, this microphotograph can be shown here only in black and white, for the micro-petrographic constituent is so characteristic in its translucent, violet-tinted shade as to allow its positive identification, whether slowly cooled from the liquid state, to permit the crystals attaining definite form, or quickly quenched, to break up the structure. Where the substance is slowly cooled, as in this case, the characteristic silicate crystallizes in long prismatic needles, with two distinct cleavages intersecting at about 90° , always biaxial and optically active.

Careful examination distinguishes an interference in the crystal formation, denoting their growth after the opaque body, carrying the greater portion of the iron oxide, had solidified. This is consistent with the reasoning of the case, as the iron oxide, having the higher melting point, would separate out first, leaving the more fusible silicate to envelop and seal up the oxide body.

Fig. 2 is a microphotograph of puddle cinder magnified 40 diameters. This magnified reproduction of a well-known material shows a formation similar to that shown in Fig. 1, but obtained at a lower temperature and more quickly cooled. This combination is formed in the liquid pig at the incipency of the puddling process, by the oxidation of the metalloid, silicon, which combines with sufficient oxidized iron from the bath to allow it to be rabbled off in a semi-liquid state.

While very few of the crystals have attained prismatic form, the predominant body has the same characteristic tint as the silicate just described. It is interspersed with the black, opaque, iron oxide, as in the heating cinder. A careful examination reveals three tiny microscopic pores, which are essentially the result of gas-bubbles entrapped in the liquid mass. Such resultant pores are always isolated and never intercommunicative, however great their number. Furthermore, inasmuch as they must remain in the liquid until it is solidified, the walls of the resultant cell will be coated with an impervious glaze of the last material to solidify.

Fig. 3, a microphotograph of Heberlein agglomerate of flue-dust, magnified 40 diameters, shows the structure of the product of heat treatment approaching nearest that of the puddle cinder just described. This agglomerate was, in all probability, made from flue-dust high in carbon, which is essential for the operation of the Heberlein process.

The crystals of silicate are coarse and well formed, showing that in this region the material was completely liquid for a short time. The proportion of the black, opaque, iron oxide constituent is large, showing that there is about a balance of the silicate sealing up the iron oxide and the iron oxide enveloping the silicate. A very few pores are evident, as there was essentially throughout its formation a current of gas passing

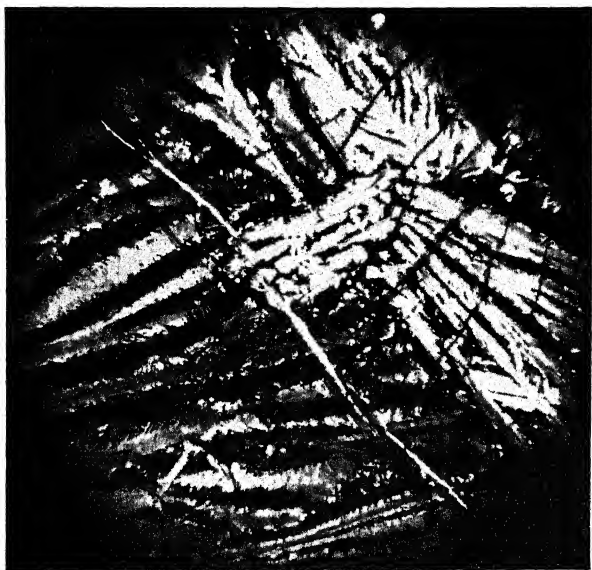


FIG. 1.—HEATING-FURNACE CINDER.

Magnified 40 Diameters. Cooled from liquid state and crystalline.
Large excess light-colored ferro-silicates. No cellular structure.

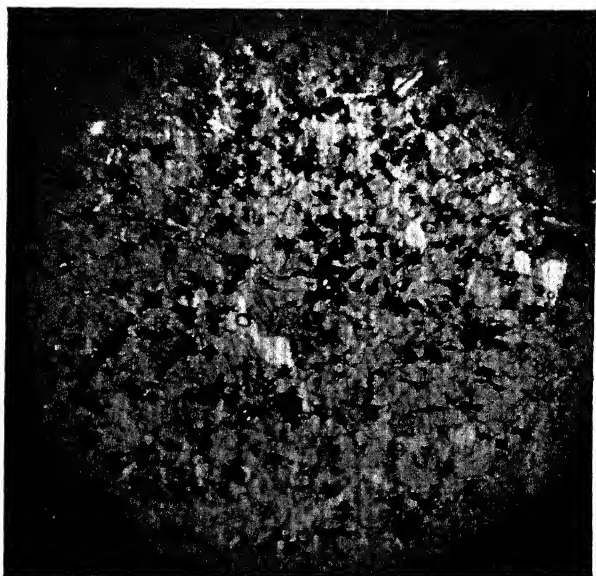


FIG. 2.—PUDDLE CINDER.

Magnified 40 Diameters. Cooled from liquid state and crystalline.
Large excess light-colored ferro-silicates. No cellular structure.

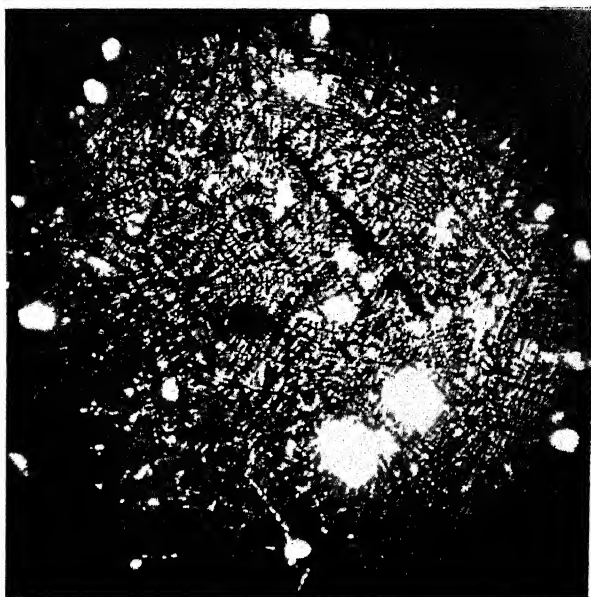


FIG. 3.—HEBERLEIN AGGLOMERATE OF FLUE-DUST.
Magnified 40 Diameters.

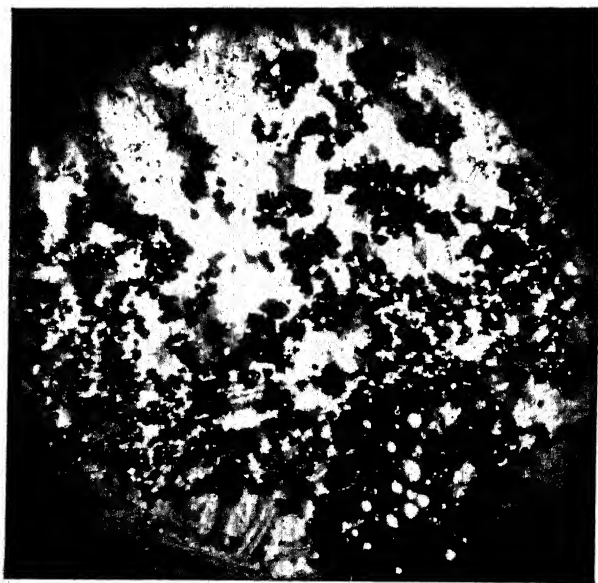


FIG. 4.—GRASSELLI CHEMICAL CO.'S HEBERLEIN AGGLOMERATE
FROM PYRITES CINDER.
Magnified 40 Diameters. Cooled after complete fusion and crystalline. Large excess light-colored ferro-silicates. Note similarity to puddle cinder. No cellular structure.

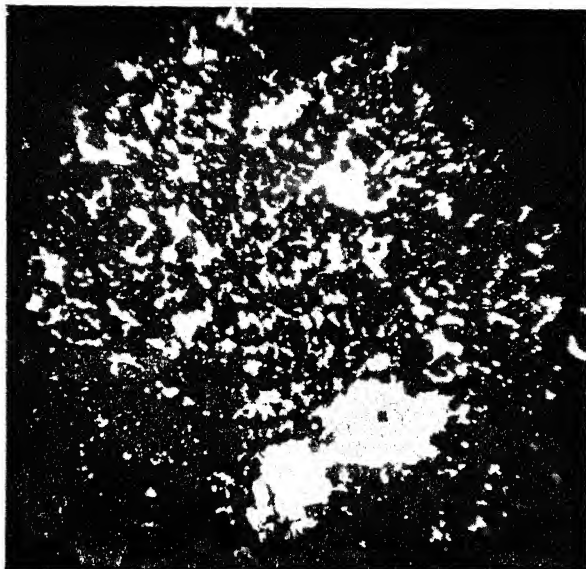


FIG. 5.—ROTARY-KILN NODULES FROM RED HEMATITE ORE. Magnified 40 Diameters. Cooled after complete fusion and crystalline. Large amount light-colored ferro-silicates. Note similarity to puddle cinder. No cellular structure.

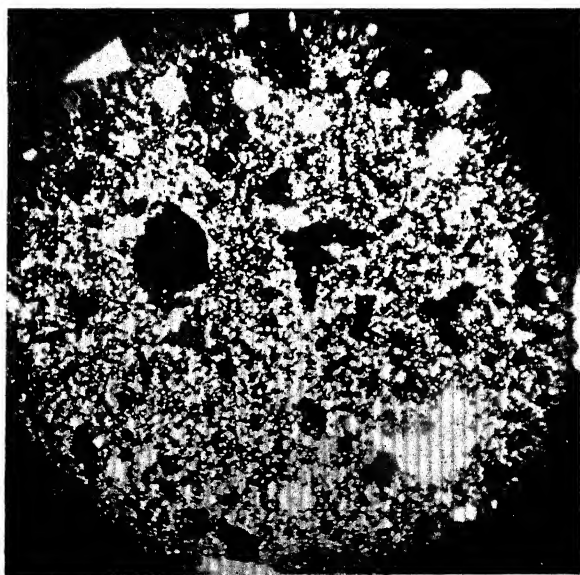


FIG. 6.—AMERICAN SINTERING Co.'s ROTARY-KILN NODULES FROM FLUE-DUST. Magnified 40 Diameters. Cooled after complete fusion and crystalline. Large excess light-colored ferro-silicates. Note similarity to puddle cinder. No cellular structure.

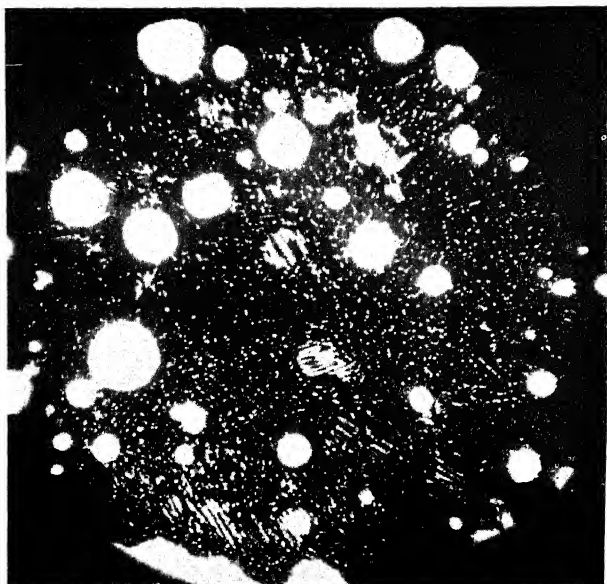


FIG. 7.—DWIGHT AND LLOYD SINTER FROM FLUE-DUST AND PYRITES CINDER.
Magnified 40 Diameters. Small amount light-colored ferro-silicates.
Large excess dark, opaque, iron oxide. Few crystals and only slightly fused. Cellular structure fully developed.

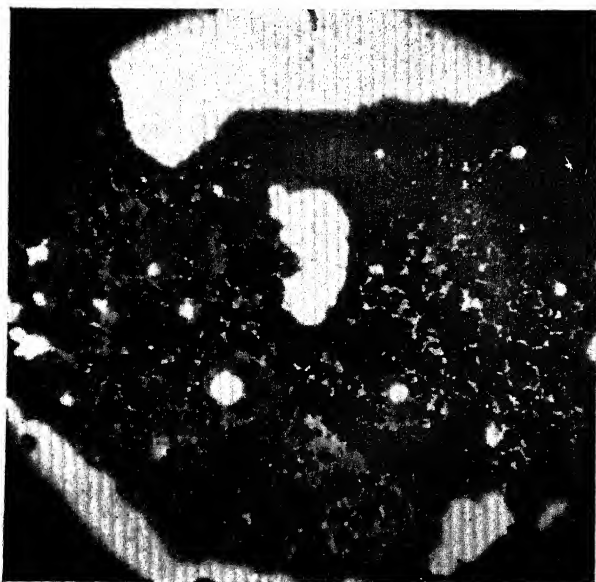


FIG. 8.—DWIGHT AND LLOYD SINTER FROM CORNWALL ORE.
Magnified 40 Diameters.

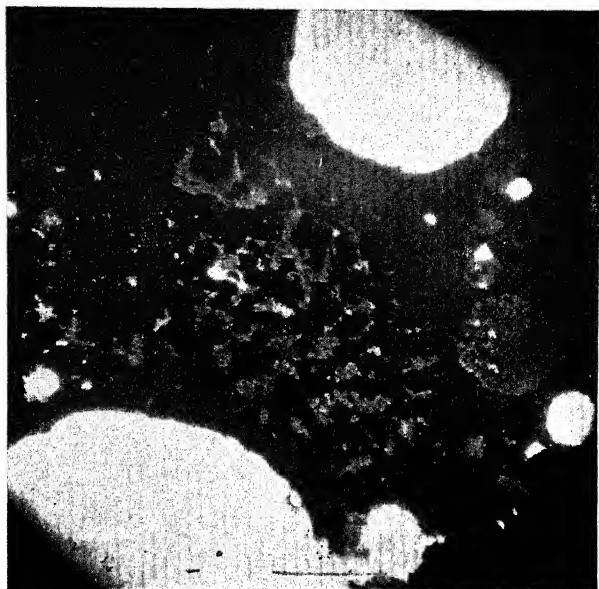


FIG. 9.—DWIGHT AND LLOYD SINTER FROM NORWEGIAN MAGNETIC CONCENTRATES. Magnified 40 Diameters.

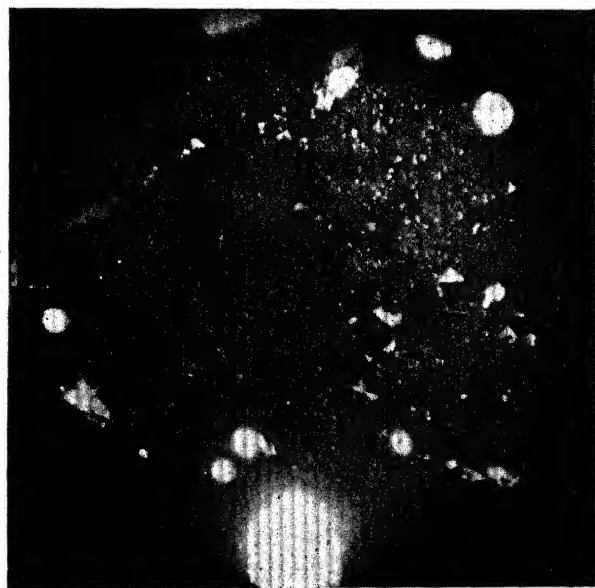


FIG. 10.—DWIGHT AND LLOYD SINTER FROM MAGNETIC CONCENTRATES. Magnified 40 Diameters.

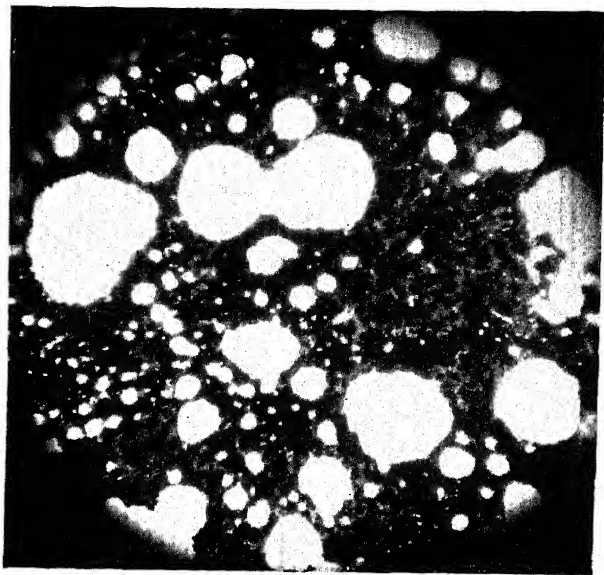


FIG. 11.—DWIGHT AND LLOYD SINTER FROM GRANULATED
HEMATITE ORE.

Magnified 40 Diameters. Traces of light-colored ferro-silicates. Larger part of mass dark, opaque, iron oxide. Few crystals and only slightly fused. Cellular spaces predominate.

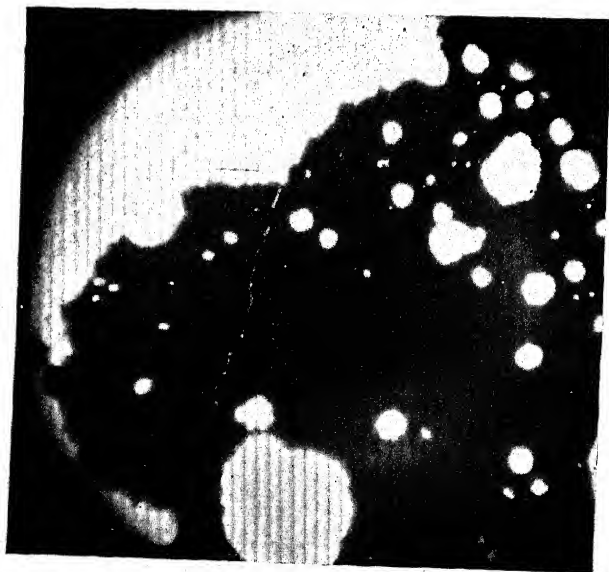


FIG. 12.—DWIGHT AND LLOYD SINTER FROM FLUE-DUST
AND MAGNETITE.

Magnified 40 Diameters. Complete absence of light-colored ferro-silicates. Whole mass dark, opaque, iron oxide. No crystals and only slightly fused. Cellular structure fully developed.

upward, which is a feature of the process. The fern-like, bi-axial crystals may be noted as having a form almost identical with those of the heating cinder. They are much smaller, due to the quicker cooling and interference from the cooling effect of the up-draft.

In macroscopic structure, this product always has cell walls many times the diameter of the cell. With such a structure, having the above-described ultimate structure, the reducibility is naturally expected to be negative.

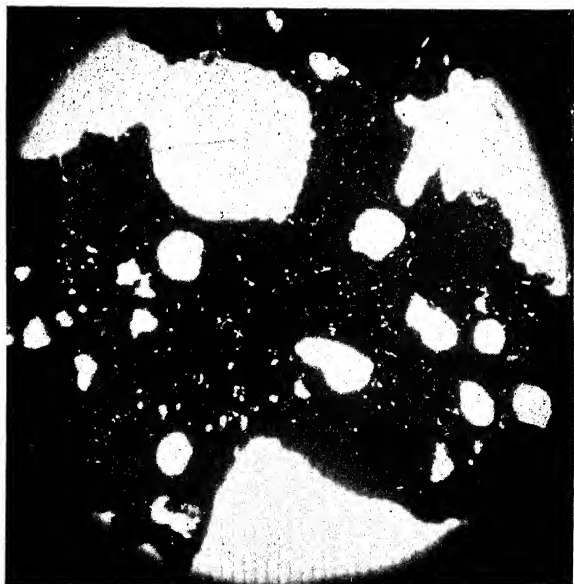


FIG. 13.—DWIGHT AND LLOYD SINTER FROM FLUE-DUST. Magnified 40 Diameters. Complete absence of light-colored ferro-silicates. Whole dark mass opaque iron oxide. No crystals and only slightly fused. Cellular structure fully developed.

Fig. 4 is a microphotograph of Heberlein agglomerate from pyrites cinder (magnified 40 diameters) as made by the Grasselli Chemical Co. This agglomerate, which was made for the double purpose of agglomerating and desulphurizing the raw material, shows very clearly the result of a high degree of fusion.

Under the microscope the specimen shows all of the area to consist of the characteristic violet-tinted iron silicate. In the lower portion of the field is seen the biaxial, prismatic crystalline structure which predominated in the heating-cinder

section. As the material is evidently higher in iron than puddle cinder, there is a greater volume of the black, opaque, oxide-bearing constituent than was seen in the puddle-cinder section. Practically no free silica is visible in the field under the microscope, showing that all the silica present had entered the iron silicate state, with its characteristic imperviousness.

In the lower right-hand portion of the above field may be noted a few pores, the result of the upward current of gases referred to in describing Fig. 3.

From the above characteristic ultimate structure, wherein the iron oxide is sealed up in a glass case of silicate, it will be readily seen that from its inherent nature such material will reach the hearth of the blast-furnace unreduced, when it forms an appreciable part of the burden, and the furnace is driven for a production approaching that usually attained with Mesabi ores. The structure is analogous to mill cinder, and similar results are to be expected from its use.

Fig. 5, a microphotograph of rotary-kiln nodules from red hematite ore, magnified 40 diameters, shows the ultimate structure of the product of the rotary kiln, working on a hematite ore. There are no cells apparent, but both the macroscopic and the microscopic structure is like that of cinder.

The crystals of the characteristic silicate structure are very evident. The one seen in the larger area near the bottom of the field has the well-developed and large biaxial crystals previously noted, which shows that the chemical constituents of this body were imprisoned in the plastic mass long enough to allow large and well-formed crystals to grow. The smaller areas of the translucent silicate are essentially less well formed, due to the interference of the solidifying iron oxide. It must be borne in mind that this is a building-up process. Free silica is practically absent, showing that all of that element present was combined with the iron to form the body referred to. It is necessary that a high temperature be maintained to nodulize the particles, and such a temperature will of necessity produce an appreciable quantity of the impervious sealing glass noticed in Figs. 1 and 4.

Fig. 6, a microphotograph of rotary-kiln nodules from flue-dust (magnified 40 diameters), as made by the American Sintering

Co., shows a large area of the silicate formation. The silicate crystals are smaller, which is very probably due, in a measure, to the cooling and disturbing effect of the chain traveling through the nodulizing mass, as is the practice in the Clarke process. Practically all the silica in the mixture of raw material is combined, as there is no free silica seen under the microscope. There is a slight tendency towards porosity in the flue-dust nodule that is absent in the ore nodule. This we attribute to the presence of the coke-dust in the flue-dust, which has its oxidation in the mass deferred to the high temperature near the discharge end of the kiln. Here the material is discharged and cooled while the resultant gases are still escaping.

While the rotary-kiln nodule as produced from flue-dust has a reducibility factor greater than that of the nodule produced from ores, it is to be expected to be a little lower than that of a Lake Superior hematite. The presence of the predominant silicate glaze of course accounts for this, and the statement is further borne out in blast-furnace practice where a portion of the burden approaching 50 per cent. of nodules was attempted.

Fig. 7 is a microphotograph of Dwight and Lloyd sinter from flue-dust and pyrites cinder (magnified 40 diameters) made from raw materials of very unusual analysis. The flue-dust, which furnished the fuel for the equal portion of pyrites cinder, was from an Eastern blast-furnace and contained the elements usually found therein. The pyrites cinder used was from a Canadian pyrites which carried 5 per cent. of BaO. The mixture therefore contained all the factors of the extraordinarily fusible tribasic slag, viz.: BaO, CaO, MgO. Although the temperature maintained in the sintering of the mixture was sufficiently low to allow very effective desulphurization,—namely: down to 0.09 per cent. of sulphur in the product,—the constituents assumed a crystalline form because of their unusual fusibility. This micro-section is herewith illustrated as being the nearest approach of a Dwight and Lloyd sinter to the microstructure of the sections already described. Hence, placed in this order, it serves as a transition member of the series.

Attention, however, is drawn to the fact that the crystalline structure has none of the violet tint of characteristic iron silicate. It is seen almost pure white or colorless under the microscope. The crystals are also all of a parallel needle structure

and are neither biaxial nor intersecting. The conclusion from the facts we have to base upon is that there is no iron compound in these crystals, but that they are all a tribasic silicate slag.

The presence of free silica is quite notable in this specimen, showing that there was insufficient heat to combine the silica and the iron present, yet enough to bind effectually the physical particles into a strong cell-wall sinter and simultaneously remove the sulphur almost completely. The cellular structure of the mass is highly favorable in that it allows a great area of contact for the reducing gases.

In all the Dwight and Lloyd products from various materials the cell structure and the characteristic shape of the cell border are notable. The cells are produced by moving currents of gas as the limited plasticity of the binding action ends. This results in intercommunicating canals, which allow the free passage of all the gases produced by the reaction. There are no isolated cells, since the method of formation forestalls them.

Fig. 8 is a microphotograph of Dwight and Lloyd sinter from Cornwall ore. This is the result of sintering Cornwall ore with 3 per cent. of fuel, having the object of desulphurization. With this extremely low amount of fuel, the ore, previously crushed to 0.25-in. size, was re-agglomerated into a sinter of very strong cell walls, and the sulphur content lowered from more than 2 per cent. in the ore to 0.08 per cent. in the sinter.

Although the material contains more than 18 per cent. of silica, there is no iron silicate shown under the microscope. The field shows a large area of some glass-like substance, but it is broken at all points by the predominating black, opaque, iron-oxide-bearing body.

The crystalline structure here occurs in grains and irregular patches, the crystals having indented faces. The small amount of silicate formed is pleochroic from brown to greenish brown. Considerable free silica is noted, which shows the control of the temperature.

The control of the temperature throughout the mass to prevent the formation of the impervious silicates, and yet to keep it sufficiently high to cause the removal of the sulphur by the blast-roast, and at the same time to produce such a highly porous product, is a feat hardly within the range of any other process than the one of Dwight and Lloyd.

Fig. 9, a microphotograph of Dwight and Lloyd sinter from Norwegian magnetic concentrates, shows practically none of the translucent silicates previously described. The silica is very low in the ore from which the sinter is made, yet some free silica is noted under the microscope. However, in order to bind the particles effectively it is necessary that the temperature of incipient fusion of the mass be approached.

Attention is drawn to the extreme porosity of the structure. The pores are carried down into the extreme microscopic range. Due to its extreme fineness and decidedly granular nature, the sintering of this material was difficult until the fine points of control were reached, then it proved quite easy. Since the material is composed of practically all iron oxide and is also very porous, its value in the blast-furnace is evident.

Fig. 10 is a microphotograph of Dwight and Lloyd sinter from magnetic concentrates, magnified 40 diameters. This product is from a magnetic concentrate similar to the one described in Fig. 9, but somewhat lower in iron and higher in silica. It is massive and very regular in structure. Its porosity is very high. There is a practical absence of any iron silicates, but some free silica is seen under the microscope. The description of Fig. 9 will apply to Fig. 10 also.

Fig. 11 is a microphotograph of Dwight and Lloyd sinter from granular hematite ore, magnified 40 diameters. This material, made from a somewhat lean ore, demonstrates the highly cellular ultimate structure characteristic of the products of the Dwight and Lloyd process. While there were sufficient slag-forming materials present to produce a glass-like coating characteristic of the more complete fusion processes, such coating in this specimen is conspicuous by its entire absence. The high-power magnification of this specimen shows no glazing or impervious coating on the walls of the pores, as is always found on the cell side of the cell walls of the products of a degree of fusion sufficiently high to maintain the slag liquid while the cells are forming. Some free silica as well as some free bases are found in this example. The predominant black, opaque, iron oxide is here seen.

A material of such extreme porosity and permeability as this, and with the absence of the silicate glass which renders the cinder-like products irreducible, is ideal for the blast-furnace,

and is evidently superior in reducible qualities to any of the ordinary hematites.

Compare this structure with that of the rotary-kiln flue-dust nodules and with that of the Heberlein pyrite-cinder agglomerate, all of which yield similar analyses.

Fig. 12 is a microphotograph of Dwight and Lloyd sinter from flue-dust and magnetite, magnified 40 diameters. This figure shows an almost complete absence of any translucent silicates. The black, opaque, massive iron oxide envelops the slag-forming material, rather than forming a chemical combination with it. Both the concentrate and the flue-dust being in an extremely fine state of division, there is almost absolute homogeneity prevailing. The cellular structure bears out the characteristic feature of the Dwight and Lloyd product. Under the microscope there can be distinguished only one constituent, and that is interspersed with innumerable pores and intercommunicating canals which render the product about the most ideal aggregation of easily reducible conditions conceivable.

Fig. 13, a microphotograph of Dwight and Lloyd sinter from flue-dust (magnified 40 diameters), shows the product of a straight flue-dust containing about 8 per cent. of carbon. The iron silicate is completely absent. The cellular structure is beautifully developed and uniformly distributed. It is desired to draw attention to the slightly irregular line forming the boundary of the section of the cell in this and all the Dwight and Lloyd products. Contrast it with the smooth, glass-like walls of the products of the processes employing the more complete fusion. This difference is obviously due to the fact that in the Dwight and Lloyd process the pores are formed by the passage of the current of gases through the mass when in a state of incipient plasticity, whereas, in the case of the other processes, the gases are entrapped in the liquid or semi-liquid glass which solidifies last, this forming an impervious cell wall.

From the foregoing observations, we logically arrive at the following conclusions:

(1) The permeability of the cell wall of a sintered product varies inversely as the degree of fusion to which it has been subjected.

(2) In a product of complete fusion, the silica present com-

bines with its equivalent of iron oxide to form a perfect glass, which, from its greater fluidity, envelops and seals up the remaining iron oxide from the action of gases.

(3) Conversely of the foregoing conclusion, in the product of the lowest degree of fusion, the iron oxide and slag-forming materials as a unit are bonded together by incipient fusion, leaving the predominant iron oxide free and vulnerable to the action of the gases in the highest degree attainable in solid products.

(4) The above salient facts show the Dwight and Lloyd products, when properly made, to possess those properties which distinguish them from the products of other sintering processes or agglomerating methods, by freedom from those constituents to which scouring action in the blast-furnace is attributed.

New Design of Open-Hearth Steel-Furnace Using Producer-Gas.

BY HERBERT F. MILLER, JR., VERONA, PA.

(New York Meeting, February, 1913.)

For a long time I have believed that the gas- and brick-costs of open-hearth furnaces using producer-gas could be greatly decreased by a change in the design of the port, which would materially reduce the first-cost of the furnace, the rebuilding-cost, and the repair-cost. The defects of the present type of such furnaces, which has done duty for so many years, may be studied in Fig. 1, in which *AA* are the two air-uptakes, *B* is the gas-uptake, and *C* the end of the gas-port.

In the present type of furnace, *C* is practically on the edge of the bath and therefore combustion begins to take place at that point, but has no effect until the gas passes from 5 to 10 ft. into the hearth.

As the furnace grows older, *C* moves back by reason of the burning or falling down of the arch. The furnace reaches its highest efficiency when *C* has moved a certain distance, but is still near enough to the bath for the overhead layer of air to hold the gas down on the bath, and far enough back to permit combustion to take place at some distance from the bath. This latter fact means that the temperature is sufficient to melt steel at the edge of the port. From this time on, the furnace decreases in efficiency. What is the reason?

As *C* moves further back, the gas is released from the gas-port at a point where there is little or no overhead layer of air. The result is that the gas floats along the roof, with the air mostly underneath it. Why the air does not stay over the gas is readily seen in Fig. 1. The lines of draft naturally run parallel to the length of the furnace. The air-uptakes being on the sides and the gas-port in the middle, most of the air will naturally be drawn along the sides of the furnace and the layer of air will be thinnest over the gas, where it is most needed.

Thus, as *C* moves back the overhead layer of air becomes thinner until *C* has reached a point where it is in a line with uptakes *AA*, and where there is no overhead layer of air. The gas then goes to the roof. The life of this type of furnace is from 180 to 250 heats.

The foregoing history of the run of producer-gas furnaces, showing the causes of loss of control of the gas, and its results in poor combustion, with consequent comparatively short run, suggests the question, Can a furnace be designed for producer-gas which will have a good gas-control and a good combustion to the end of a run of, say, from 400 to 700 heats?

This end can be attained by either of two designs, both of which involve the same principle of putting a single air-uptake directly back of the gas-uptake, as is shown in Figs. 2, 3, 4 and 5.

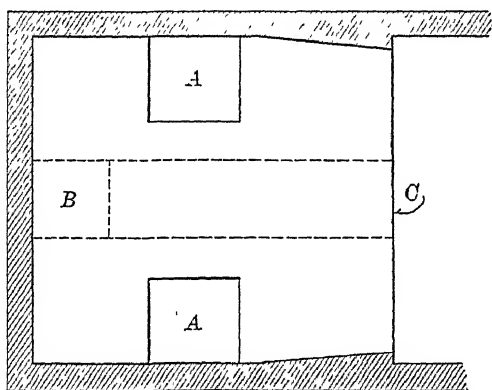


FIG. 1.—PRESENT TYPE OF FURNACE.

In Figs. 2 and 3 the port has about one-third of the width of the hearth, whereas the present type has the full width. The roof of the port dips sharply, compressing the air down on the gas; and the roof of the hearth is gradually dropped to meet it.

The gas-port arch is retained, although flattened to the width of the port. The end *C* is from 6 to 10 ft. from the edge of the bath.

There is a small seal-door on each side of the port at *C* to permit the repair of the port or the removal of the brick which might fall from the gas-port arch.

The point *C* will move back slower than in the present type,

because it will be less exposed to the flame, which will always be under the arch.

The possible defect of this design is that when the gas-port arch has become short combustion will commence too far from the bath, and make a high gas-cost. Another objection is, that this design could not be applied to existing furnaces because their ports would have to be lengthened considerably. If the

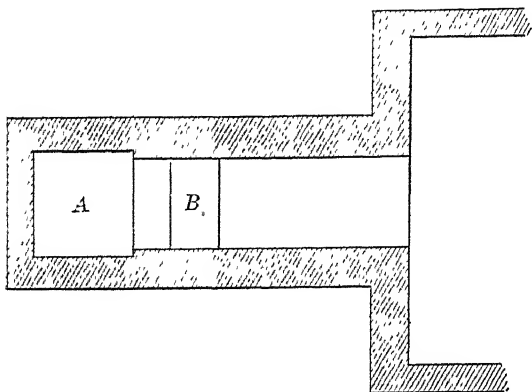


FIG. 2.

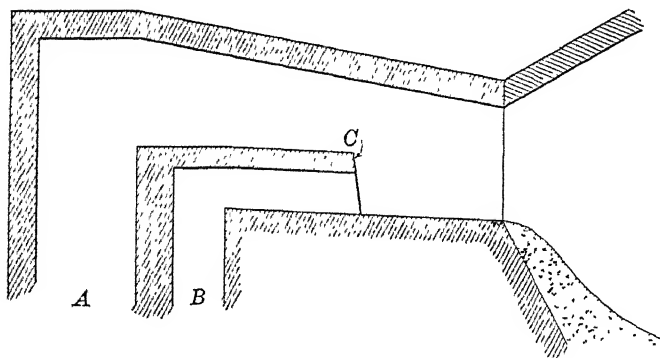


FIG. 3.

FIGS. 2 AND 3.—IMPROVED TYPE OF FURNACE.

flame in the above design can always be controlled, even when the gas-port arch has burned back to the gas-uptake, why have a gas-port arch at all?

The design shown in Figs. 4 and 5 is the solution. Here it will be seen that the furnace is similar in outline to the present

open-port type in use at Homestead. The air-uptake *A* is back of the gas-uptake, from which it is separated by a substantial wall. The gas-uptake, which is as wide as the port, is from 6 to 10 ft. from the bath. The port is open and smooth, narrow and low. The small cross-section of this port assures a speedy union of the preheated air and gas, resulting in a short flame of intense heat, which will be under control throughout the run.

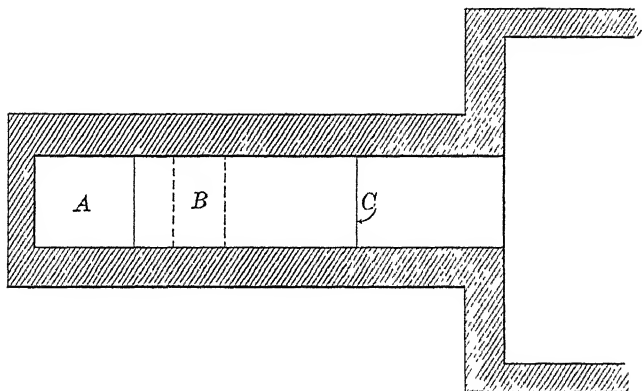


FIG. 4.

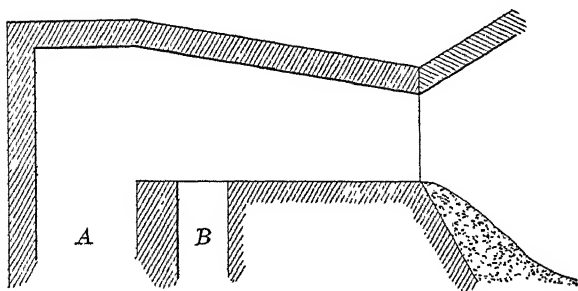


FIG. 5.

FIGS. 4 AND 5.—ALTERNATIVE DESIGN OF FURNACE.

Some may think that the gas rising at right angles to the air will make a high flame. If the port is low enough this will not occur. I have put a stream of air having a pressure of 70 lb. per sq. in. under a natural-gas flame, and found that there was very little deflection of the flame.

This design could be applied to the present type of producer-gas furnaces. The only alteration of importance would be a

reversal of the gas-checkers from the outside to the inside position.

In building a new furnace, a great saving of brick would be made in the ports, as can be readily seen.

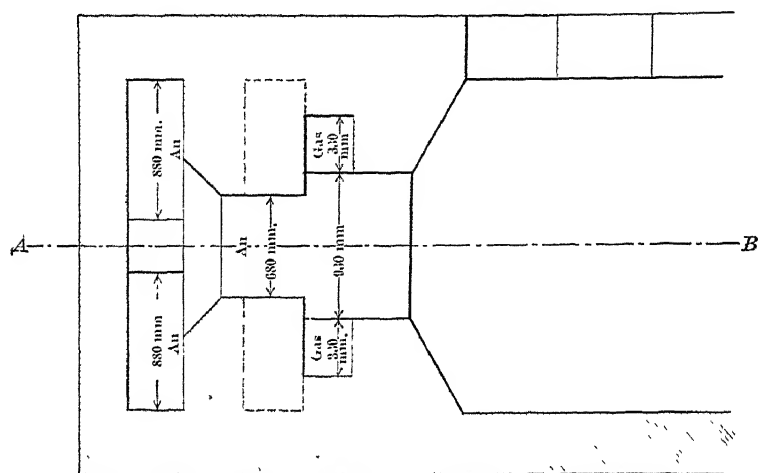


FIG. 6.

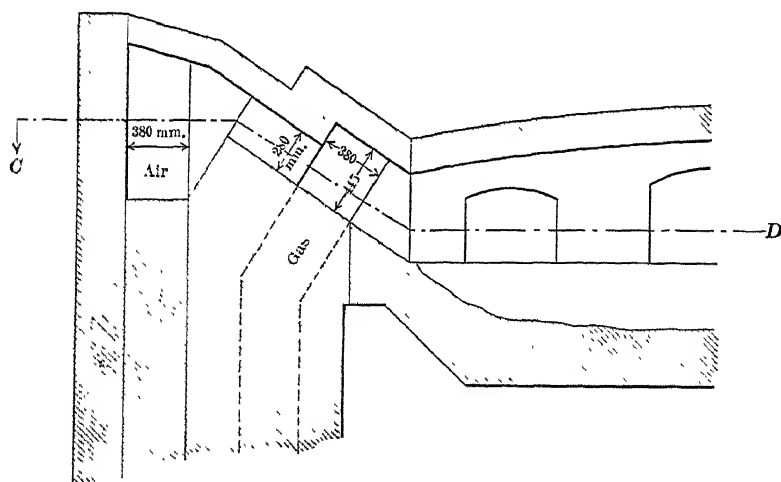


FIG. 7.

Gas-flues $= 2 \times 330 \times 330 = \text{about } 0.250 \text{ sq. m. area.}$

Air-flue $= 630 \times 280 = \text{about } 0.175 \text{ sq. m. area.}$

Mixing-flue $= 930 \times 445 = \text{about } 0.415 \text{ sq. m. area.}$

FIGS. 6 AND 7.—DESIGN OF SWEDISH OPEN-HEARTH FURNACE.

Repairs to the dividing-wall can be made through small seals by the side of the gas-port. The flame will always be under

control, and the furnace should slow up only because of dirty checkers and insufficient draft. Natural gas could be used if desired by introducing it, as usual, at the sides and in front of the uptake *C*.

In my judgment, the adoption of this design would do away with much of the trouble and expense attending the present type of producer-gas furnace, and greatly decrease the operating-cost.

Finally, the flame-temperature of producer-gas being slightly higher, when the air and gas are the same temperature, than the flame-temperature of natural gas when it is burned with air at the same temperature as the air and gas in the producer-gas type of furnace, the producer-gas type of open-hearth furnace, given the same chance for combustion that the open-port natural-gas furnace has, should and will be the faster furnace.

I shall be glad to answer any questions or criticisms that may be made.

I am indebted to Joseph Hartshorne for the accompanying sketches, Figs. 6 and 7, of a furnace used in Sweden, which is similar in principle to the design described above. This furnace was operated for 58 weeks without repairs. This was in 1900, and the furnace was still operating in 1902.

DISCUSSION.

HENRY D. HIBBARD, Plainfield, N. J.:—Looking at the illustrations, it seems to me that a furnace made with the width of port that is shown here would not be a good working furnace. The economy of brick and gas might be all right, but I do not think it would melt the steel properly. I should say that the flame would not spread out suitably to fill the whole hearth of the furnace, and there would be undue waste, and it seems to me that it would be an undesirable furnace to adopt.

The question as to whether a producer-gas flame would lie down in the furnace, hugging the bottom better than the natural-gas flame, is settled, not so much by the specific gravity of the gases themselves, as by the relative temperatures at which they enter the laboratory of the furnace. We know in that case that producer-gas would be lighter than a natural gas which had not been run through a regenerator and expanded

four or five volumes in consequence. I would like to hear what Mr. Miller has to say about the action of the flame spreading properly over the charge to give good metallurgical results in the form of furnace which he presents.

HERBERT F. MILLER, JR., Verona, Pa. (communication to the Secretary *) :—The paper was written to cause a discussion of open-hearth furnace design, to the end that the gas- and brick-costs might be reduced and the production increased. The question before us is whether or not the old type of Siemens-Martin furnace, which has been practically unchanged since the open-hearth was first in use, can or cannot be improved upon. Many say that it cannot be improved. If this is so, then there is no need for this paper or any further discussion of open-hearth furnace design. I believe that the furnace can be made of a more suitable design and designed along more natural and rational lines.

Let us discuss some of the objections raised and some of the comparisons of the two types of furnace.

Some say that there is much more gas used in producer-gas practice than in natural-gas practice. We know that this is so when measured per ton of steel produced, but we know also that a given size of stack will not draw any more waste gases per minute out of a producer-gas furnace than it does out of a natural-gas open-hearth furnace. Furthermore, the hearth is the same size and the space that the flame takes up is the same in each type of furnace. The temperature of the producer-gas flame is from 300° to 500° C. hotter than that of the natural-gas flame. Therefore we have two factors balanced, and a third in favor of the producer furnace.

The next and most important factor is, the opportunity that each furnace gives for a good combustion of the respective fuels; a comparison will be interesting.

The natural-gas furnace gives its fuel (the more inflammable of the two) a chance to get in working shape before it reaches the hearth. It is then ready to melt or give heat to a molten bath and does so until it leaves the furnace as a waste product. Even in a natural-gas furnace the tendency to chill is apparent unless steps are taken to overcome this trouble.

* Received Apr. 9, 1913.

Now, in a producer-gas furnace, the gas is not in shape to do any work until it has passed over the hearth for some distance; and even then, the distance from the overhead source of combustion, the large space in which the air and gas unite, and the comparatively slow draft under which they unite, all tend to make a very poor combustion and a lower flame temperature. This poor combustion is apparent from the appearance of the gas as it leaves the port, from the design of the furnace, and from the fact that the bath freezes under the gas and at its sides as it comes in and has to be melted again when the gas is reversed, thus doing the same work over and over until the furnace has picked up enough heat in the checkers and the bath has become hot enough to overcome the chilling tendency of the entering gas.

These facts all point out that, if we give the producer-gas the same opportunity to get in shape for its work that we do its more inflammable rival, we shall have the faster furnace, because of the higher temperature that producer-gas is capable of producing; but the producer-gas furnace is, at present, the slower furnace of the two and will remain so until it is designed along more rational lines.

Objections have been raised to the proposed design of producer-gas furnace in that it is feared that the gas will rise up through the overhead air and burn the roof. Now let us review the factors present. We have a port of small cross-section, which will insure a tremendous draft; and we have gas coming up out of the uptake at a pressure of not more than 4 oz. per sq. in.; and, lastly, the body of air is heavier per cubic foot than the producer-gas. The gas does not rise at right angles to the bottom of the port, because of the powerful draft, but is immediately drawn towards the furnace with not much, if any, rise from the face of the port. Furthermore, the gas is hit by an overhead current of heavier air and is still more depressed, and will enter the furnace low and go out the same way.

By the time the flame reaches the bath it is in condition to do work, and the tendency to sulphurize the unmelted steel should be overcome by the same influences.

Also, when the calorific value of natural gas is compared with that of producer-gas, the vast difference of the two gases

in calories per unit volume so much impresses itself upon us that we forget that in practice we do not deal with cold gas from the producer, but instead have to deal with a very highly-heated fuel, and in this condition producer gas will have the advantage over natural gas; because, in melting or working a charge of steel, it will surround or cover the charge with a hotter flame, provided that both kinds of gas are given the same opportunity for combustion. When natural gas is used in a producer-gas furnace it is not subjected to the same treatment that is given to producer-gas, for preheated air is rising through the gas-uptake and combustion of the natural gas begins to take place where the pipe is introduced into the end of the furnace. If the gas-uptake be shut off, permitting no combustion to take place until the gas reaches the bath, thus giving the natural gas the same treatment which the producer-gas has, then it will be decisively demonstrated that even natural gas needs an opportunity to get into working condition before reaching the bath, because the cold entering natural gas also will freeze the bath.

If, then, each flame occupies the same space in the furnace, and each is given an opportunity to work efficiently, then we should have no more waste with producer-gas than with natural gas, and we ought to have even less. This being the case, there should certainly follow a large reduction in the amount of coal used per heat or per ton of steel produced.

Notes on Cast-Iron.

BY ALBERT SAUVEUR,* CAMBRIDGE, MASS.

(New York Meeting, February, 1913.)

It is delightful to read a technical paper like that of J. E. Johnson, The Effect of High Carbon on the Quality of Charcoal-Iron, presented in October, 1912, at the Cleveland meeting of the American Institute of Mining Engineers.¹ The clear, simple, and straightforward manner in which he describes his experiments and results makes you feel as if you had been, so to speak, at his elbow during the course of his investigations, while his method of procedure is so rational that you can fairly anticipate each step in the logical sequence of the tests performed. The following notes suggested by Mr. Johnson's paper are presented here in the hope that they may help, if only a mite, towards the final solution of a problem of considerable scientific and industrial importance.

Physical Properties of Cast-Iron vs. Its Ultimate Composition.

Mr. Johnson writes that "an investigation along the lines of consumption showed that different irons have different characters totally independent of their analyses," and he declares that "the presence or absence of these elements [meaning carbon, phosphorus, silicon, and manganese] alone will not account for all, scarcely for a half, of the facts which have long been known." He further tells us that some foundries have "proved, by the most irrefutable tests, that they could take a certain iron of a given analysis and produce certain results," while with "another iron of the same analysis they could not produce these results at all."

I feel confident that every experienced and thinking metallurgist will readily believe Mr. Johnson's statements and will share his views in regard to the hopelessness of inferring the

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¹ *Trans.*, xliv., 314 (1912).

physical properties of an iron casting from a knowledge of its ultimate composition. In 1890 Professor Howe sounded a timely note of warning for those in danger of being entangled in the snarls of ultimate composition when he wrote :

“If these views be correct, then, no matter how accurate and extended our knowledge of ultimate composition, and how vast the statistics on which our inferences are based, if we attempt to predict mechanical properties from them accurately, we become metallurgical Wigginses.”

Steel-metallurgists were prompt to realize the truth and importance of Professor Howe's contention, and strenuous efforts were made by them to obtain a knowledge of the proximate composition of steel not only as affected by ultimate composition but also by thermal and mechanical treatments. How much metallography has contributed towards this end is now a matter of common knowledge.

Blast-furnace- and foundry-metallurgists, on the contrary, have shown considerable indifference towards the irrefutable facts mentioned above, with the result that they are now confronted with unsolved problems for which solutions might possibly have been found long ago had they realized sooner the wisdom of applying scientific methods to the study of cast-iron. Many, not to say most, are still attempting to perform the feat of predicting the physical properties of castings from a knowledge of the percentages of silicon and sulphur they contain. We should, however, be fair, and when we say with Mr. Johnson that mechanical properties cannot safely be deduced from ultimate composition, we should recognize the fact that in foundry parlance chemical composition too often means nothing more than certain percentages of sulphur and silicon, or, at best, of sulphur, silicon, phosphorus, and manganese, carbon-determinations being the exception and not the rule. And still it is of course primarily upon the percentage of combined carbon it contains that the properties of a casting, such as strength and softness, depend. Certainly, a closer relation exists between its properties and its content of combined carbon than between those same properties and the percentages of all the other impurities put together, namely, graphitic carbon, silicon, manganese, sulphur, and phosphorus. Why, then, are not determinations of combined carbon more frequent?

Why, indeed, are they not invariably made? Clearly because we have no simple, quick, and accurate laboratory method for the determination of combined carbon in cast-iron. It is well known that in order to obtain an approximate knowledge of the percentage of combined carbon two rather long and tedious carbon-determinations are needed, namely, determinations of total and of graphitic carbon. The one is then subtracted from the other, and, putting our faith in the accuracy of both, results are reported supposed to represent the percentage of combined carbon. If we bear in mind that as little as 0.1 per cent. of combined carbon may affect enormously the physical properties of cast-iron, the utterly unsatisfactory character of this method "by difference" is obvious. Indeed, foundrymen recognize the hopelessness of the procedure, and seldom call for carbon-determinations, relying instead upon silicon- and sulphur-contents to guess at the probable percentage of combined carbon formed under certain cooling-conditions. This is a distressing situation, for which a remedy should be found. Dr. Drown once said that the analytical chemist was asked the impossible and did it. May we not again turn to him in our great need and ask that he perform another seemingly impossible task? The chemist who will devise a simple, quick, and accurate method for the direct determination of combined carbon in cast-iron will be entitled to the everlasting gratitude of foundry-metallurgists.

I believe that if the analysis of cast-iron always included determinations of combined and graphitic carbon the discrepancy between properties and composition would be less glaring. But there is another and very important factor affecting the properties of cast-iron which is almost invariably ignored, namely, the shape and mode of distribution of the graphite particles. At this juncture the chemist necessarily fails us, and we must turn to the metallographist for assistance. It would be folly to assume that a certain proportion of graphite always produces the same effect, regardless of the size, shape, and distribution of the particles in which that element occurs. To those who would hesitate to admit this fact we only have to point out the enormous difference in properties existing between gray cast-iron and malleable cast-iron, two metals which may conceivably be of identical composition. In the one instance,

large, curved sheets of graphite, thoroughly breaking up the continuity of the mass, and producing weakness and brittleness; in the other, small, rounded graphite particles, leaving to the iron matrix much of its strength and ductility. These different occurrences of graphite are well illustrated in Figs. 1 and 2.

An accurate knowledge of the size, form, and mode of distribution of the graphite would be a decided step forward in binding more closely together properties and composition. This is a problem for the metallographist to solve. He should find a method of ascertaining and of reporting the dimensions and forms of the graphite particles. There is no doubt but that, other things being equal, the finer and less angular and sharp the graphite particles, the stronger must be the iron. Whether or not it will ever be possible so to control the casting-operation that graphite will be made to occur in small, globular particles, as it does in malleable castings, thereby securing much greater strength, is in the present stage of our knowledge impossible to foretell, but it would be unwise to set the problem aside as an insoluble one. In 1900 I published a short note in which I showed the marked influence of the rate of cooling upon the form assumed by the graphite particles.¹ The photomicrographs illustrating this point, reproduced in Figs. 3 and 4, show the structure of the same cast-iron cooled respectively (1) slowly in sand and (2) more quickly in chilled mold. The quicker cooling resulted in the formation of rounded, star-like particles of graphite, which undoubtedly must have been accompanied by greater strength. Finally, when considering the chemical composition of cast-iron we should remember that Dr. Moldenke and others contend that the probable presence of varying amounts of oxides in the metal should be taken into account, as they may explain, in part at least, results seemingly at variance with the chemical composition as commonly reported.

Cast-Iron of Maximum Strength.

As first pointed out by Professor Howe, cast-iron should be regarded as a metal composed of a steel matrix, and of graphite

¹ The Microstructure of Coke and Charcoal Pig-Iron, Albert Sauveur. *The Metallographist*, vol. iii., p. 154 (1900).



FIG. 1.—GRAY CAST-IRON FREE FROM COMBINED CARBON. MAGNIFIED 100 DIAMETERS. NOT ETCHED. (F. C. Langenberg in the Author's Laboratory.)

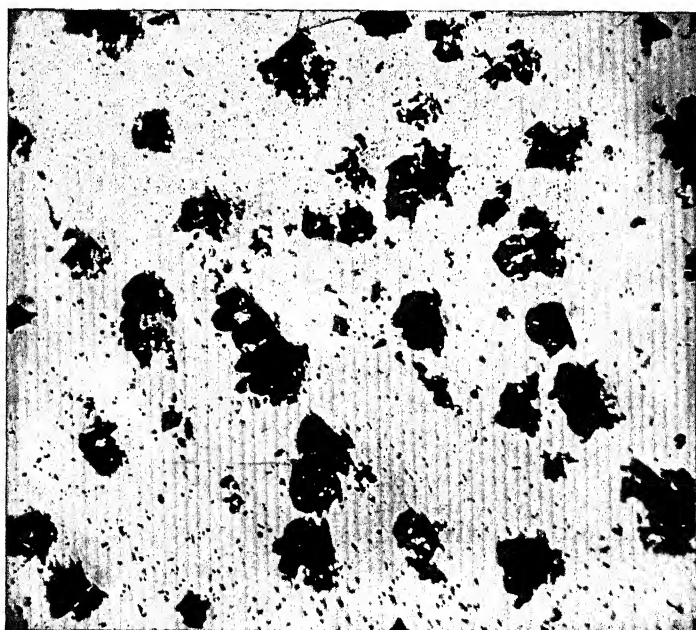




FIG. 3.—COKE PIG-IRON CAST IN SAND.
MAGNIFIED 56 DIAMETERS.



FIG. 4.—SAME PIG-IRON AS IN FIG. 3.
CENTER OF CHILLED BAR. MAGNIFIED
56 DIAMETERS.

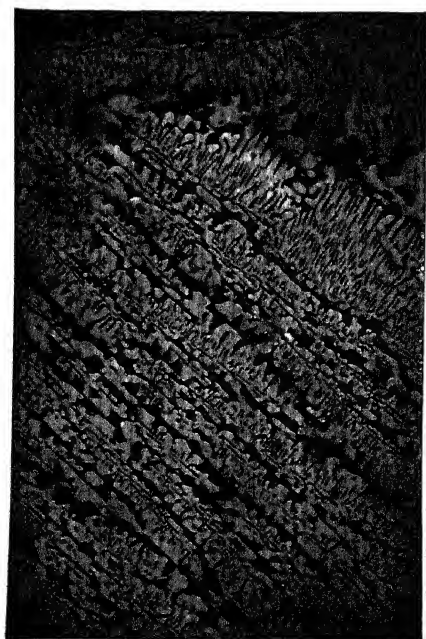


FIG. 7.—WHITE SPOT IN GRAY CAST-IRON. MAGNI-
FIED 100 DIAMETERS. (J. E. Johnson, Jr.)



FIG. 8.—AUSTENITE-CEMENTITE EU-
TECTIC. MAGNIFIED 750 DIAME-
TERS. (Goerens.)

particles. Clearly, therefore, cast-iron of maximum strength should have a matrix of maximum strength, and since eutectoid or slightly hyper-eutectoid steel is the strongest steel, cast-iron of maximum strength should have a eutectoid or slightly hyper-eutectoid matrix, that is, it should contain some 0.80 per cent. of combined carbon. It is also evident that once the desired matrix is secured (through rate of cooling, silicon-content, etc.), the fewer the graphite particles the better, which means that the lower the total carbon the better. This explains why strong cast-iron can be made more readily from metal melted in an air-furnace or, with the assistance of steel scrap, in a cupola-furnace. In melting pig-iron alone in a cupola-furnace, however, we have little control over the total carbon, which necessarily fluctuates between 3 and 4 per cent., according to silicon-content and other conditions. The shape and mode of distribution of the graphite particles should, however, receive closer attention, as they certainly influence very decidedly the physical properties. Given two cast-irons having eutectoid matrix and identical chemical composition, but in which the graphite occurs respectively in sharp, large plates and in rounded, star-like particles, the latter will undoubtedly be the stronger of the two. Can we cause the graphite to assume the more desirable form? I have already called attention to the beneficial influence, in this respect, of relatively quick cooling during and below solidification. When a casting solidifies there are two factors, which regulate more than anything else the formation of the desired amount of combined carbon, namely, its silicon-content and the rate of cooling. It is universally known that in a small casting a larger proportion of silicon is required in order not to exceed the desired amount of combined carbon, because of its quicker solidification (due to its small dimensions), promoting the formation of combined carbon. Exactly the same proportions of combined and of graphitic carbon may be produced in a small casting through high silicon and relatively quick cooling, as in a large casting through lower silicon-content and slower cooling, but it does not by any means follow that these two castings will have the same structure and hence the same properties. In the smaller casting, because of its more rapid cooling, the graphite particles will be smaller and less sharp; hence it will be the stronger of the two.

The combined action of the rate of cooling and of the silicon-content in determining the character of the matrix of cast-iron may advantageously be shown graphically as in Fig. 5. The abscissas correspond to percentages of silicon, the ordinates to rates of cooling represented by arbitrary numbers, 2 indicating quicker cooling than 1, 3 quicker cooling than 2, etc. Any point on the line AB corresponds to a set of conditions producing a eutectoid matrix, while any point above AB refers to rates of cooling and silicon-contents resulting in hyper-eutectoid, and below AB , in hypo-eutectoid matrix. The point M , for instance, on AB corresponds to 1 per cent. of Si

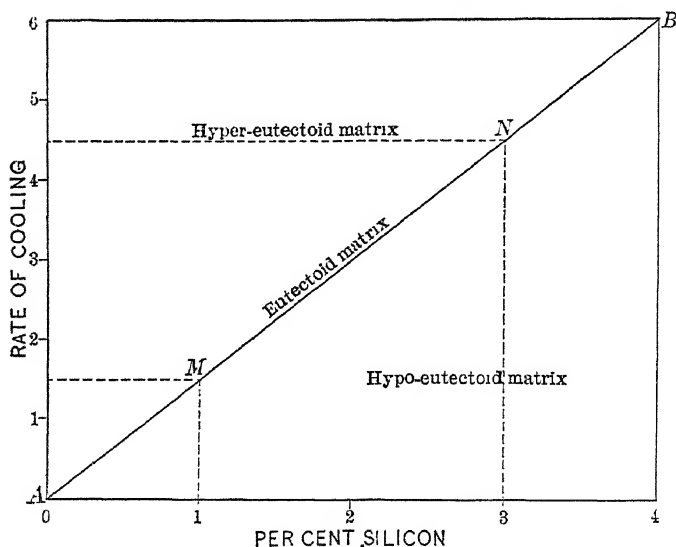


FIG. 5.—DIAGRAM ILLUSTRATING THE COMBINED ACTIONS OF RATE OF COOLING AND SILICON-CONTENT IN DETERMINING THE CHARACTER OF THE MATRIX OF CAST-IRON.

and a rate of cooling represented by 1.5, while N corresponds to 3 per cent. of Si and a quicker rate of cooling, 4.4. Both sets of conditions will produce castings having a eutectoid matrix but not identical properties, because the quicker cooling in the latter case must produce smaller and less angular graphite particles and hence a stronger metal.

If the above reasoning is correct, to produce cast-iron of great strength we should cause the formation of a eutectoid matrix by hastening its cooling in presence of considerable silicon.

Proximate Composition of Cast-Iron.

I contended long ago that a knowledge of the proximate analysis of cast-iron should be of much greater value than a knowledge of its ultimate analysis, and I believe that this view is now quite universally held. Why do we not then make an attempt at least to report proximate rather than ultimate compositions? The proximate constituents of commercial cast-iron are: (1) elemental carbon or graphite; (2) iron carbide, Fe_3C ; (3) iron silicide, FeSi ; ² (4) iron phosphide, Fe_3P ; (5) manganese carbide, Mn_3C ; and (6) manganese sulphide, MnS ; ³ and the relative proportions of these constituents may readily be calculated from a knowledge of the ultimate composition. Basing this calculation upon the atomic weights of the elements present in the different chemical compounds, we find that the percentage of silicon multiplied by 3 gives the percentage of FeSi ; that the percentage of phosphorus multiplied by 6.4 gives the percentage of Fe_3P ; that the percentage of sulphur multiplied by 2.5 gives the percentage of MnS ; that the percentage of MnS multiplied by 0.63 gives the percentage of Mn present as MnS which should be subtracted from the percentage of Mn in the cast-iron and the difference multiplied by 1.07 in order to obtain the percentage of Mn_3C ; that the percentage of Mn_3C multiplied by 0.07 gives the percentage of carbon in Mn_3C which should be subtracted from the total amount of combined carbon in the cast-iron and the difference multiplied by 15 in order to obtain the percentage of Fe_3C .

Let us assume, for instance, a cast-iron of the following ultimate composition :

	Per Cent.
Graphitic carbon.....	3.00
Combined carbon.....	0.50
Silicon	2.00
Phosphorus	1.00
Manganese.....	0.40
Sulphur	0.03
Iron by difference.....	93.07
	<hr/> 100.00

² It is believed by some that FeSi_2 , rather than FeSi is present in cast-iron.

³ I purposely ignore the possible presence of FeS in the absence of a sufficient amount of manganese to combine with all of the sulphur.

Its proximate composition will be

	Per Cent.
2×3	= 6.00 FeSi
1×6.4	= 6.40 Fe ₃ P.
0.03×2.5	= 0.07 MnS.
$0.07 \times 0.63 = 0.04, 0.40 - 0.04 = 0.36, 0.36 \div 1.07$	= 0.38 Mn ₃ C.
$0.38 \times 0.07 = 0.03, 0.50 - 0.03 = 0.47, 0.47 \div 15$	= 7.05 Fe ₃ C.
	3.00 graphite.
	77.10 iron by difference.
	<hr/> 100.00

The Structural Composition of Cast-Iron.

The following structural constituents of commercial cast-iron are readily distinguishable under the microscope: graphite, free cementite, free ferrite, pearlite, and the phosphide eutectic for which I proposed long ago the name of "steadite" in honor of J. E. Stead, its discoverer. Particles of MnS when sufficiently large and abundant may likewise be easily detected under the microscope, but may be ignored in calculating the structural composition because of the extremely small volume occupied by them. The iron silicide FeSi (or, according to some, FeSi₂) does not occur as a separate constituent, but is apparently dissolved in ferrite, while the carbide Mn₃C unites with Fe₃C to form cementite. Bearing these facts in mind, the structural composition of any cast-iron may readily be calculated if we know the percentages of graphitic carbon, combined carbon, and phosphorus it contains.

Stead has shown that the iron-phosphide eutectic, or steadite, contains very nearly 10 per cent. of phosphorus, from which it follows that in multiplying by 10 the percentage of phosphorus in the iron we obtain the percentage of steadite. We should also bear in mind that 15 times the combined carbon gives the amount of total cementite. It is necessary to ascertain first whether the cast-iron considered will have a hypo-eutectoid, eutectoid, or hyper-eutectoid matrix. The following considerations will make this possible: Let G represent the percentage of graphitic carbon, C the percentage of combined carbon, and Ph the percentage of phosphorus. In order to have a eutectoid matrix that is to be free from both free ferrite and free cementite, the metal must contain these two constituents in the proportion of 7 to 1, and since $100 - G - 10Ph - 15C$ evi-

dently represents the percentage of total ferrite, and 15C that of total cementite, we must have

$$\begin{aligned}\text{Ferrite} &= 7 \text{ cementite, or} \\ 100 - G - 10\text{Ph} - 15\text{C} &= 7 \times 15\text{C} = 105\text{C, or} \\ 100 - G - 10\text{Ph} - 120\text{C} &= 0\end{aligned}$$

for cast-iron having a eutectoid matrix. If the first term of the equation is greater than 0 the metal will contain free ferrite, *i. e.*, its matrix will be hypo-eutectoid; if smaller than 0 it will contain free cementite, *i. e.*, it will be hyper-eutectoid. In the presence of 3 per cent. of graphite, for instance, and 1 per cent. of phosphorus the percentage of combined carbon needed to produce a eutectoid matrix will be obtained in solving the equation:

$$100 - 3 - 10 - 120\text{C} = 0$$

which calls for 0.75 per cent. of combined carbon.

If the cast-iron has a hypo-eutectoid matrix its structural composition will be obtained through the resolution of the following equations:

$$\begin{aligned}(1) \text{ P} + \text{F} + \text{S} + \text{G} &= 100 \\ (2) \text{ P} &= 120\text{C} \\ (3) \text{ S} &= 10\text{Ph},\end{aligned}$$

in which P, F, S, and G represent, respectively, the percentages of pearlite, ferrite, steadite, and graphite. The first equation expresses the fact that the cast-iron considered is made up of the above constituents; the second, that the percentage of pearlite is obtained in multiplying by 120 the proportion of combined carbon; and the third, that steadite equals ten times the phosphorus-content.

Taking, for instance, the cast-iron already considered, we find its structural composition to be

	Per Cent.
Graphite.....	30
Pearlite.....	60
Steadite.....	10
Free ferrite.....	27
	<hr/> 100

If the cast-iron has a hyper-eutectoid matrix the following equations will permit the ready calculation of its structure:

$$\begin{aligned}(1) \text{ P} + \text{C} + \text{S} + \text{G} &= 100 \\ (2) \text{ P} &= \frac{7}{8} (100 - 15\text{C} - 10\text{Ph} - \text{G}) \\ (3) \text{ S} &= 10\text{Ph},\end{aligned}$$

the first one expressing the fact that the metal is made up of pearlite, cementite, steadite, and graphite; the second, that the percentage of pearlite is $\frac{2}{3}$ that of the total ferrite; and the third, that steadite is ten times the phosphorus.

Spots of White Cast-Iron Surrounded by Gray Iron.

Passing to the interesting phenomenon of the formation of spots of white cast-iron surrounded by gray iron, Mr. Johnson first rejects as unsatisfactory the explanation ascribing it to the pressure exerted upon the still molten center by the shrinking solid shell, a pressure which prevents the formation of graphitic carbon since such formation is accompanied by increased bulk. He rejects it on the ground that although iron of all grades cools under precisely these conditions, only a small percentage of it is spotted. Later in his paper, however, he offers this very explanation to account for the white spots of his iron. That it is the true cause of the formation of white-iron centers seems highly probable and a little reflection will show that the absence of white centers in most pig-iron is in keeping with it. There must be a period during the solidification of any iron casting when its still liquid interior is surrounded by a solidified shell. At that instant and during its solidification and further cooling this internal portion of the casting is subjected to three influences controlling the formation of graphite: (1) the presence of silicon promoting graphite-formation; (2) slow cooling (the metal being now protected by the red-hot shell), which also promotes the formation of graphite; and (3) the pressure exerted by the now contracting shell, which opposes the formation of graphite. It is not surprising that in most cases influences (1) and (2) prevail and that graphite is formed in spite of influence (3). It is only when influence (3) is greater than (1) and (2) combined that white spots will be produced. For the same rate of cooling, therefore, which generally implies the same size of castings, white centers should be formed the more readily the less silicon in the iron, and for the same silicon-content they should form more readily in small castings.

Percentage of Carbon in White Spots.

In regard to the percentage of carbon in the white spots, it is certainly logical to infer with Mr. Johnson that it should

show a tendency to approach the eutectic ratio, since clearly these white spots correspond to the metal solidifying last, and it is natural to assume that the most fusible constituent—that is, the eutectic alloy—must show a tendency to migrate towards that portion of the casting remaining liquid longest. When Mr. Johnson talks of the iron-carbon eutectic, however, ascribing to it a carbon-content of 4.30 per cent., clearly he refers to the iron-carbon diagram, which is true only for pure iron-carbon alloys, whereas Mr. Johnson's samples are ternary alloys of iron, carbon, and silicon, not to mention the other impurities. The occurrence of nearly 0.5 per cent. of silicon, which is the amount present in some of Mr. Johnson's spotted samples, has

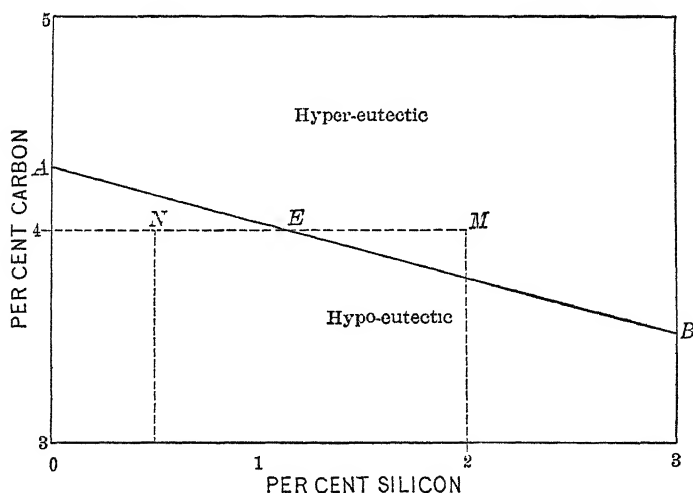


FIG. 6.—DIAGRAM ILLUSTRATING THE INFLUENCE OF SILICON ON THE PERCENTAGE OF CARBON IN THE IRON-CARBON EUTECTIC.

a marked influence upon the carbon-content of the eutectic. From the work of Wüst and Petersen⁴ we are led to infer that the presence of 1 per cent. of silicon lowers that carbon some 0.30 per cent. These authors' results may profitably be represented graphically as shown in Fig. 6. The meaning of the diagram is obvious. The line *AB* divides cast-iron into hyper- and hypo-eutectic metal according to its percentages of carbon and silicon. With 4 per cent. of carbon and 2 per cent. of silicon, for instance, the cast-iron is hyper-eutectic, since its composition is represented by the point *M* in the

⁴ *Metallurgie*, vol. iii., No. 24, p. 811 (Dec. 22, 1906).

hyper-eutectic range, while with the same amount of carbon but in the presence of only 0.5 per cent. of silicon, the metal is hypo-eutectic, since its composition is represented by the point *N* in the hypo-eutectic range, etc.

Solidification of Eutectic Cast-Iron.

The fine photomicrographs published by Mr. Johnson, especially his Fig. 5 (reproduced in Fig. 7 of the present paper), certainly tend strongly to support the theory that the white spots must be of eutectic composition. It should be borne in mind, however, that while these structures exhibit the characteristic features of eutectic alloys they no longer represent such alloys, because of the structural changes having taken place on cooling from the solidification to atmospheric temperature. They have, however, preserved enough of the structural characteristics of eutectic alloys to show clearly that they solidified as such. To make this matter clear let us recall the mechanism of the solidification and subsequent cooling of the iron-carbon eutectic alloy when no graphite is formed. On reaching its solidification-point ($1,130^{\circ}$ C. in the case of pure alloys) crystalline plates are formed alternately of cementite and of austenite (a solid solution of iron and of the carbide Fe_3C , containing about 1.7 per cent. of carbon, or 25.5 per cent. of Fe_3C). On further cooling the solid solution constituent rejects some of its cementite until it retains but some 0.85 per cent. of carbon (12.75 per cent. of cementite). By this time the temperature of the alloy has reached its Ar_1 point and the remaining austenite breaks up into pearlite, itself a eutectoid alloy of iron (ferrite) and Fe_3C . These important changes, however, leave the structure of the metal sufficiently undisturbed to make it evident that a eutectic alloy formed on solidification. Clearly, however, Mr. Johnson's photographs do not represent the eutectic alloy itself. The dark constituent of his structure is not austenite containing some 1.7 per cent. of carbon, as it should be were it a compound of the eutectic alloy, but it is pearlite, that is, the ferrite-cementite eutectoid. Examination of his samples under sufficiently high magnification to resolve pearlite should convince him of the accuracy of the statement. Mr. Johnson refers to a photomicrograph taken by Goerens, which I reproduce in Fig. 8, and indeed some of his illustra-

tions bear strong resemblance to it. But Goerens actually shows the structure of the eutectic, which he preserved by quick cooling immediately after solidification, while in Mr. Johnson's the original cementite plates have absorbed the cementite rejected by the austenite, and the latter has finally been converted into pearlite because of slow cooling after solidification.

Following a scheme I originated long ago and now quite commonly used, the mechanism of the solidification of eutectic cast-iron and of its subsequent transformation on slow cooling may be illustrated graphically as shown in Fig. 9, assuming that no graphite forms. *ABCD* represents a block of eutectic cast-iron cooling from above its melting-point to atmospheric temperature. Above its liquidus-solidus line *EF* the metal is liquid. On reaching that line, which in pure iron-carbon alloys corresponds to a temperature of 1,130° C., it solidifies as an austenite-cementite eutectic containing *EG* (47.7 per cent.) of austenite and *GF* (52.3 per cent.) of cementite.⁵ On cooling from the solidus to the eutectoid line *HL*, or *Ar₁* point, at about 700° C., the eutectic austenite rejects cementite (pro-eutectoid cementite) until it reaches the eutectoid composition (0.85 per cent. of C or 12.75 per cent. of Fe_3C). The pro-eutectoid cementite thus formed joins the eutectic cementite and is represented by the triangle *GIK* in the diagram. Clearly *IL* represents the total free cementite in the alloy when it reaches its eutectoid point. On cooling through this point the remaining austenite, now of eutectoid composition, breaks up into ferrite and cementite to form pearlite, a constituent which contains about 12.75 per cent. of cementite (*OI* in the diagram) and 87.25 per cent. of ferrite (*HO* in the diagram). Finally, then, the completely-cooled alloy contains *ND* per cent. of cementite of eutectic origin, *MN* per cent. of cementite of pro-eutectoid

⁵ The composition of the eutectic is readily ascertained, since we know the percentage of carbon in cementite (6.67 per cent.) and in eutectic austenite (1.7 per cent.). It suffices to solve the two equations:

$$\begin{aligned} (1) \quad A + C &= 100; \\ (2) \quad \frac{1.7}{100} A + \frac{6.67}{100} C &= 4.30; \end{aligned}$$

the first one expressing the fact that the eutectic contains austenite and cementite; the second, the fact that the carbon in the austenite plus the carbon in the cementite is equal to the total carbon present (4.30 per cent.).

origin, PM per cent. of eutectoid cementite and CP per cent. of ferrite. The eutectic and pro-eutectoid cementite known collectively as "free" or "excess" cementite (MD) are so merged that they cannot be distinguished from each other, while the eutectoid cementite exists as a distinct constituent of pearlite. In the diagram PD represents the total cementite (free and eutectoid) in the alloy and CP the ferrite; that is, its ultimate structural composition.

Solidification of Hyper-Eutectic Cast-Iron.

Mr. Johnson's explanation of the mechanism of the solidification of hyper-eutectic cast-iron violates the rigid rules governing the constitution and formation of eutectic alloys in general as well as the phase rule. He argues that when hyper-eutectic cast-iron reaches its solidification-point graphitic carbon solidifies, *i.e.*, falls out of solution, while at the eutectic temperature a eutectic alloy is formed composed of austenite and cementite. This, of course, cannot be. If it is graphite that forms when the iron begins to solidify, graphite also must be a component of the eutectic, and the latter should be an austenite-graphite and not an austenite-cementite eutectic. The fact that an austenite-cementite eutectic is undoubtedly formed proves that it is cementite and not graphite that formed above the eutectic temperature. If graphitic carbon of pro-eutectic origin is found in the iron, it must have been formed through the breaking up or dissociation of pro-eutectic cementite while cooling from the liquidus to the solidus. Those who still believe that under certain conditions graphite may form directly from molten iron realize, of course, that this must imply the formation of an austenite-graphite eutectic. The opposite view, however, which holds that carbon is dissolved in molten iron as the carbide Fe_3C , that it always solidifies as such, breaking up subsequently into graphite and iron when the conditions are favorable (slow cooling, presence of silicon, etc.), is steadily gaining ground and is better supported by experimental evidences. The mechanism of the solidification of hyper-eutectic cast-iron and of its subsequent transformation on slow cooling may be represented graphically, as shown in Fig. 10. It is assumed in this diagram that no graphitic carbon is formed.

On reaching its liquidus, EF , in the vicinity of $1,200^{\circ}$ C. in

the case of a pure alloy containing 5 per cent. of carbon, cementite begins to form, and its formation continues to the solidus line, as indicated by the triangle FGH . On reaching the solidus ($1,130^{\circ}$ C.) the molten portion of the alloy has the eutectic composition (4.30 per cent. of carbon), and it solidifies as an austenite-cementite eutectic, IG , containing IK per cent. of austenite and KG per cent. of cementite. On cooling from its solidus to its eutectoid or A_1 point (700° C.) the eutectic austenite rejects cementite, as indicated by the triangle KQN , until, on reaching the eutectoid point, it is of eutectoid composition, and is converted into pearlite containing LM per cent. of ferrite and MQ per cent. of cementite. The alloy will finally consist of UD per cent. of pro-eutectic cementite, TU per cent. of eutectic cementite, ST per cent. of pro-eutectoid cementite, RS per cent. of eutectoid cementite, and CR per cent. of ferrite. The pro-eutectic, eutectic, and pro-eutectoid cementites are merged, and cannot be distinguished from each other under the microscope. Their sum, SD , constitutes the free cementite of the alloy. The eutectoid cementite, RS , on the contrary, remains intimately associated with the ferrite to form the constituent pearlite, represented by CS in the diagram. RD represents the total percentage of cementite in the alloy.

Solidification of Hypo-Eutectic Cast-Iron.

The mechanism of the solidification of hypo-eutectic alloys may likewise be graphically represented as shown in Fig. 11. On reaching its liquidus, EF , pro-eutectic austenite containing 1.70 per cent. of carbon begins to form. Assuming the cast-iron to contain 3.50 per cent. of carbon, its temperature would then be in the vicinity of $1,225^{\circ}$ C. In cooling from the liquidus to the solidus or eutectic temperature, GK , austenite continues to form, the percentage of that constituent solidifying between the liquidus and the solidus being represented by the triangle EGH . On reaching the eutectic temperature, the liquid portion of the alloy has reached the eutectic composition and now solidifies as an austenite-cementite eutectic, HK , containing HI per cent. of austenite and IK per cent. of cementite. On cooling from the eutectic to the eutectoid or A_1 temperature, both the pro-eutectic and the eutectic austenite reject cementite, as indicated by the triangle IMN . At the eutectoid tem-

perature, LO , the remaining austenite is of eutectoid composition and is transformed into pearlite, LM , containing LT per cent. of ferrite and TM per cent. of cementite. Below the eutectoid temperature, therefore, the cast-iron consists of CR per cent. of pearlite and RD per cent. of free cementite, the pearlite containing CQ per cent. of ferrite and QR per cent. of cementite, and the free cementite RS per cent. of cementite of eutectic origin, and SD per cent. of cementite of pro-eutectoid origin. QD evidently represents the total cementite in the alloy.

The Graphitizing of Cementite.

It is well known that cementite readily breaks up into iron (ferrite) and graphite when heated to a sufficiently high temperature, and it is likewise well known that the higher the temperature the more easily does this dissociation take place, and that silicon also promotes it. Bearing in mind the influence of temperature on the graphitization of cementite, we are justified in assuming that the higher the temperature at which cementite forms the more readily will it be converted into ferrite and graphite on subsequent cooling. It follows from the foregoing assumption that the graphitization of the cementite formed in iron-carbon alloys during and below their solidification will take place in the following order with increasing difficulty: (1) graphitization of pro-eutectic cementite, (2) graphitization of eutectic cementite, (3) graphitization of pro-eutectoid cementite, and (4) graphitization of eutectoid cementite.

Graphitizing of Hyper-Eutectic Alloys.

In hyper-eutectic alloys pro-eutectic cementite forms as soon as the alloy begins to solidify and keeps on forming to the eutectic temperature. At such high temperature the dissociation tendency is so great that graphite very readily forms, even in the absence of silicon and during relatively rapid cooling. If time be given, the graphite resulting from the dissociation of pro-eutectic cementite rises to the surface of the still liquid bath, when it is known as "kish." Upon reaching the eutectic temperature, eutectic cementite forms, which easily breaks up into iron and graphite if the cooling be sufficiently slow, its graphitization being assisted by the presence of *nuclei* of pro-eutectic graphite. In cooling from the eutectic to the eutectoid

temperature pro-eutectoid cementite is expelled by the austenite. The graphitization of pro-eutectoid cementite, however, proceeds with less readiness because of the relatively low temperature now prevailing, although assisted by the presence of many *nuclei* of pro-eutectic and eutectic graphite. A large percentage of silicon and very slow cooling would of course promote it. If some of the pro-eutectoid cementite fails to be graphitized the resulting cast-iron will necessarily have a hyper-eutectoid matrix and will probably be mottled. Upon reaching the eutectoid temperature eutectoid cementite is formed, but its graphitization, although greatly assisted by the presence of much graphite, will require, to be complete, at least very slow cooling and the presence of considerable silicon. It is, of course, evident that if some pro-eutectoid cementite has escaped graphitization none of the eutectoid cementite will graphitize, since the dissociation of cementite becomes more difficult as the temperature is lowered. In case the graphitization of eutectoid cementite is complete, a rare instance, the resulting cast-iron has a pure, or rather carbonless, ferrite matrix. If it is incomplete the matrix will be hypo-eutectoid and the cast-iron belong to one of the foundry grades.

Graphitizing of Eutectic Alloys.

On reaching the solidification temperature eutectic cementite is formed, which breaks up quite readily into ferrite and graphite unless prevented from doing so through rapid cooling. On cooling from the eutectic to the eutectoid temperature pro-eutectoid cementite forms, the graphitization of which is promoted by the presence of *nuclei* of eutectic graphite but opposed by the lower temperature now prevailing. It may, however, be complete in the presence of sufficient silicon. If it remains incomplete the cast-iron will have a hyper-eutectoid matrix. The graphitization of the eutectoid cementite formed at the eutectoid temperature is much less readily brought about, being now opposed by the low temperature of the metal. It will generally remain incomplete and the resulting cast-iron will have a hypo-eutectoid matrix.

Graphitizing of Hypo-Eutectic Alloys.

In hypo-eutectic alloys when solidification begins austenite is formed and at the eutectic temperature eutectic cementite

falls out of solution. This cementite undergoes graphitization quite readily because of the high temperature at which it forms. On cooling from the eutectic to the eutectoid temperature pro-eutectoid cementite is rejected by austenite, and its graphitization proceeds the more slowly that it is not now assisted by pre-existing *nuclei* of graphite to the same extent as eutectic and hyper-eutectic alloys. The eutectoid cementite is graphitized with difficulty because of low temperature and of relatively little pre-existing graphite. If the graphitization of pro-eutectoid cementite remains incomplete the cast-iron will have a hyper-eutectoid matrix, while if the graphitization of eutectoid cementite remains incomplete it will have a hypo-eutectoid matrix. In case of complete graphitization of the pro-eutectoid cementite and of no graphitization at all of the eutectoid cementite the cast-iron would have a eutectoid matrix.

Behavior of Cementite in Steel.

Hyper-eutectoid steel solidifies as a solid solution, austenite, which on cooling from the $A_{r_{cm}}$ point to the eutectoid point (A_{r_1}) rejects pro-eutectoid cementite. Because of the low temperature now prevailing and in the absence of *nuclei* of graphite this cementite is not dissociated. For similar reasons the eutectoid cementite formed at a still lower temperature remains undecomposed.

In the case of hypo-eutectoid steel eutectoid cementite only is formed and necessarily, for similar reasons, remains undecomposed. In steels, therefore, because of their low carbon-content, which precludes the formation of cementite at a high temperature, none of the cementite can be decomposed, and they are consequently free from graphitic carbon even after very slow cooling and in the presence of much silicon. A small amount of graphitic carbon may be formed in high-carbon steel, however, by prolonged annealing at high temperature.

According to the phase rule, when the dissociation of the cementite is complete, so that the metal finally contains only the two phases ferrite and graphite, its equilibrium is stable; if no graphite forms, the two phases present being ferrite and cementite, its equilibrium is metastable; while if the graphiti-

zation of the cementite is incomplete, the metal containing the three phases, ferrite, cementite, and graphite, its equilibrium is unstable. It follows from the foregoing considerations that white cast-iron and slowly-cooled steel are in a metastable equilibrium, that gray cast-iron is very generally in unstable equilibrium, and that malleable cast-iron generally, and gray cast-iron very rarely, are in stable equilibrium.

DISCUSSION.

Including discussion of the paper of J. E. Johnson, Jr., *The Effect of High Carbon on the Quality of Charcoal-Iron*, *Trans.*, xlv., 314.

CHAIRMAN JOHN BIRKINBINE:—Just 20 years ago I had the pleasure, when I retired from the Presidency of the Institute, of introducing my successor, and this afternoon it is my pleasure to ask my good friend Dr. Howe to open this discussion.

PROFESSOR HENRY M. HOWE, New York, N. Y.:—This paper of Mr. Sauveur's is extremely interesting. It is one of those papers that one wishes to read in the quiet of one's study, where it can be read with great profit. There are many points which he makes which I think are of very great value, in particular as to the importance of getting fine-grained graphite by a relatively rapid cooling, combined with a high percentage of silicon. That theory would seem to be very promising; how far it has actually been carried out very likely Dr. Moldenke can tell us.

I think Professor Sauveur is quite right in taking exception to the common saying that you cannot tell anything about the properties of cast-iron from its composition. It is very true that you cannot from the composition which is generally given us, because that composition usually leaves out the most important and essential thing, that is to say, the quantity of combined carbon; that is not generally given to us. It is more important than anything else in the cast-iron, because the other elements—the manganese, the silicon, and the sulphur—act in very large part in determining how much combined carbon there shall be, and it is that resultant, the percentage of combined carbon, which is of importance rather than the things which determine the resultant, namely, the quantity of silicon, manganese, and sulphur.

Some years ago we used to say that we could not tell very certainly about steel from its composition. We certainly cannot tell with perfect accuracy, so much depending on the manipulation it has been through. Cast-iron is still more complicated, because it has all the complications of steel, the base of the cast-iron being a steel matrix, added to which are the variations due to the condition of the graphite which is present.

Mr. Johnson seemed to recognize, and Professor Sauveur recognizes very clearly, that the essential thing, the most prominent and important thing, in the composition of cast-iron is its relation to the eutectic ratio; that is to say, whether it is hyper-eutectic or hypo-eutectic. A discussion of that involves some familiarity with the iron-carbon diagram.

By means of this diagram we may be able to bring about an understanding of the relations between the composition of cast-iron and its properties.

The presence of oxygen has been always one of the ways by which we have tried to explain what we could not otherwise explain, and oxygen has always been used as the thing on which to throw the blame.

I must say, with all due deference to Dr. Moldenke's acuteness and knowledge, that it seems difficult to suppose that a cast-iron which has come out of the blast furnace, in which the deoxidizing power has been so great that it has deoxidized the silicon, as it does to so large an extent, still contains iron oxide which has been able to run that gauntlet and persist in the cast-iron. (We should remember that the deoxidizing conditions seem to reach their greatest intensity at the very end of the blast furnace process, when the molten iron lies in the crucible of the furnace overlain by a layer of slag, and with the end of the column of white hot coke extending down through that slag into the metal itself. Here the deoxidizing action of the coke is not lessened, as it is above, by the only mildly deoxidizing atmosphere of the furnace. It is here that the deoxidation of silicon reaches its maximum. Under these conditions I find it difficult to believe that oxygen can remain in the molten iron, and there is no later favorable opportunity for the absorption of oxygen, unless in the remelting in the cupola furnace.) [These words in parentheses were added by correspondence.]

I do not say it cannot happen, because many things we say cannot happen we find afterwards do happen, but I only say it is one of those things which are so improbable antecedently that it requires the strongest sort of evidence to make us accept them.

The diagram which Professor Sauveur has given on p. 358 is of very great usefulness, and he does a very useful task in calling our attention to the importance of silicon in determining the relation to the eutectic ratio, *i. e.*, in determining whether the iron is hypo- or hyper-eutectic.

I question whether Mr. Johnson is mistaken in the way in which Professor Sauveur says he is on p. 367.

On p. 368 Professor Sauveur, I think, indicates that the three forms of cementite are necessarily merged in cast-iron and cannot be distinguished. Fig. 12 is a micrograph in which we have great flakes of the primary or pro-eutectic cementite, which are totally distinct from the matrix, and in that matrix you can find the pro-eutectoid and eutectoid cementite, apparently, quite distinct from the pro-eutectic cementite.

As regards distinguishing the eutectic cementite from the pro-eutectoid cementite, I think that sometimes that can be done, but I am not sure.

It is true, as Professor Sauveur says, that graphitization does not readily become thoroughly complete, yet we do find some pig-irons, commercial pig-irons, in which the graphitization is complete—in which there is no combined carbon. All of the cementite—the pro-eutectic, the eutectic, the pro-eutectoid and the eutectoid—seems to have graphitized completely. The evidence leaves hardly any doubt that it is true. (See *Trans.*, 1908, p. 516.)

Upon p. 371 Professor Sauveur speaks of the freedom of steel from graphite; that is to say, the cementite of steel does not readily graphitize. That is true of most steels, but when you get the carbon high enough, as in file steel, then the cementite does graphitize to a rather surprising extent. In fact, that is one of the things that, in making file steel, have to be guarded against. The production of black or graphitic file steel seems to be due to internal release of pressure in rolling or hammering, which enables the cementite to graphitize.

Professor Sauveur says (on p. 372), that the equilibrium in

gray cast-iron is unstable. That expression has to be taken in the Pickwickian sense. It is unstable from the point of view of the physical chemist, but it actually has a very high degree of stability, and probably would persist forever without any change. We know that the Japanese sword blades of the fifteenth century are as hard on their edges now as though they had been freshly hardened. Those edges are very hard because of the martensite which is present, and the martensite is very much more unstable than any cementite in gray iron. If that martensite will persist without any decomposi-

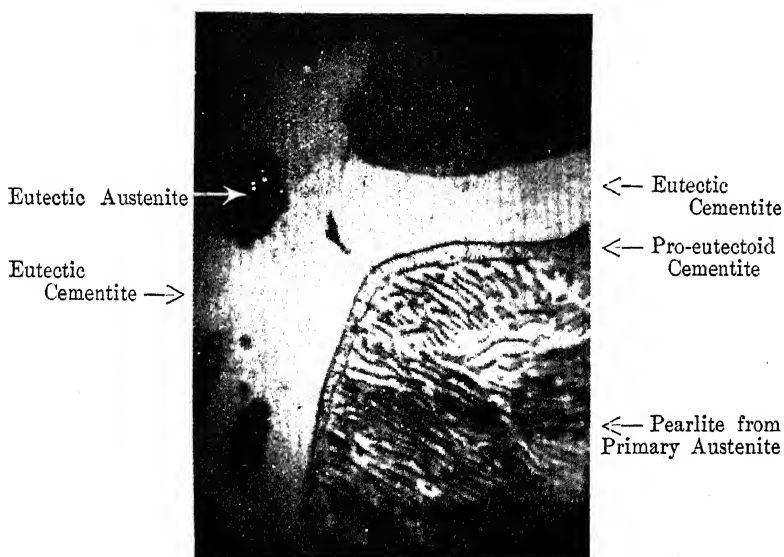


FIG. 12.—REPRODUCTION OF BENEDICKS'S FIG. 12, *Metallurgie*, vol. iii., No. 13⁹ following p. 432 (July 8, 1906). Magnification 1,000.

tion for hundreds of years, I think from that, and from our general knowledge of the subject, we may assume that cementite in pig iron under ordinary conditions and at atmospheric temperature would remain undecomposed forever.

THE CHAIRMAN:—Is there any further discussion?

PROF. HOWE:—There is one point I omitted. In the diagrams on p. 366, which are very beautiful and very instructive, I think the diagonal straight lines should be curved.

(Since the meeting at which Professor Sauveur's paper was discussed, I learn from Sir Robert Hadfield that his specimen of iron from the Delhi Column, which contains about 0.08 per cent. of carbon, has "no trace of graphite.")

If, as is probable, this column dates from about 300 A. D., it follows that cementite can endure at least for a period of about sixteen centuries without graphitizing. One would suppose that the temperature of the Delhi Column must at times be pretty high, if it is exposed to the sun of India.

The Ovifak mass of native iron contains both pearlitic and pro-eutectoid cementite. If, as is probable, this mass dates from an earlier geological age, we infer that cementite, though nominally metastable, in fact can resist decomposition forever.)

[The paragraphs in parentheses were added by correspondence.]

DR. RICHARD MOLDENKE, Watchung, N. J.:—It is a somewhat difficult matter to discuss Professor Sauveur's paper, as so much depends upon the point of view. I have been so constantly on the firing line in the foundries of the country that I view the matters presented more from the standpoint of practical experience with large quantities of molten iron than from laboratory investigations.

Professor Sauveur evidently assumes that cast-iron, *per se*, is far more uniform in structure than it really is. Conclusions drawn from the structure of the metal of the interior of a casting cannot be applied to that of the surface of the same casting, as the two are entirely different in every case.

I do not agree with Professor Sauveur in his contention that the chemical composition of cast-iron is of little value because it has been found that it does not give a safe basis upon which to predict the characteristics of a casting. Quite the contrary; unless the chemical composition is correct for a given purpose, failure is certain to result. Melting and other conditions, if not properly cared for, may prevent the realization of expectations, but a proper chemical composition is essential as the foundation of the work in hand.

The principal cause of failure to realize a definite result with a given composition of cast-iron, to my mind, lies in the point that Dr. Howe raised—namely, the supposed oxidation of the

iron during the process. Whereas he questions this occurrence, I firmly believe in it, and have so believed for years, as it seems to offer the only explanation for the troubles encountered. In the making of pig-iron, as Dr. Howe rightly states, the iron runs the gauntlet of a series of reducing influences, and consequently it should be free from dissolved iron oxides. This I question, for while the iron sponge formed by the reduction of the ore may be perfectly free from oxidation, when this sponge gets down lower and melts it is in a region where there is free oxygen present about the tuyeres. Depending upon the fuel used and the temperature and pressure of the blast, this free oxygen will extend inward to a greater or less extent, and affect the melting iron sponge. To my mind, this very thing is the cause of the difference in strength and other qualities between pig-irons of identical composition when made with cold blast charcoal, hot blast charcoal, and very hot blast coke as the blast-furnace fuel. Remelting such pig-irons in the cupola or air furnace in foundry practice—all these being oxidation and not reduction processes—simply accentuates the difficulties which make for uncertain results.

Professor Sauveur is correct in assuming that combined carbon must be looked after in castings to get the required results. While it is well known that the percentage of combined carbon present—and this varies considerably from surface to interior of the casting—is the result of influence upon the total carbon present of silicon, manganese, sulphur, phosphorus, the casting temperature, and the rate of cooling due to the sections, the unknown quantity of dissolved iron oxide present (0.3 per cent. being sufficient to ruin the iron for casting purposes absolutely, as proved by actual determinations) is still more powerful for evil. Sulphur, to which nearly all our troubles have been ascribed, and which is a bad thing to have around, is still safer to have in quantity than iron oxide. I have made castings with 0.22 per cent. of sulphur which machined perfectly, whereas ordinarily 0.10 per cent. is pretty bad. But they were made of iron melted carefully and not subjected to oxidizing influences in the cupola, as far as this could be done. I may state that in my consulting practice I have been called to hundreds of foundries where they were getting defective castings, and rarely did I have to change the mixture. It was simply

the readjustment of the melting conditions that had to be looked after, so that melting in the zone of free oxygen was avoided.

Professor Sauveur calls for a good method of determining combined carbon. The difficulty for foundry purposes is to make use of such a method. The combined carbon varies from surface to interior of the casting, and the castings are apt to be made before the metal can be tested for this ingredient. However, the foundry laboratory will gladly welcome such a method, as the present method of ascertaining it by subtraction of graphite from total carbon is tedious and expensive.

As to the question of high silicon and great strength when cooling is hastened, I agree with Professor Sauveur; in actual practice this idea is carried out for such castings as pipe balls and glass molds. In fact, where it is possible to keep the silicon high enough to compel the formation of graphite, and where artificial cooling is resorted to so that the graphite crystals will be very small, and further, where sufficient steel scrap is added to reduce the total carbon considerably, the strength will be high, and the casting remain soft.

Professor Sauveur's separation of the ultimate and rational analysis ideas, so far as cast-iron is concerned, is excellent, and while not of special use in daily practice, is nevertheless of the highest interest in studying out the possibilities of the material. The percentages of the combinations as shown by the microscope give an illuminating insight into the structure of the material, which it is not possible to obtain from the ultimate analysis. The difficulty, for daily foundry practice, lies in the fact that a casting once made cannot be subjected to heat treatment as readily as is the case with steel.

Finally, the question arises of white spots in iron surrounded by the gray mass. I am inclined to agree with Professor Sauveur in this and hold that it is entirely a matter of pressure. In drawing conclusions from observations made in casting iron it must not be forgotten that some castings have their interior under pressure while the metal is setting and others have not—simply as the result of the presence or absence of dissolved gases, or, if you will, oxides in solution. Dr. Howe this morning spoke of "puffing up" steel before casting into ingots, as a means for preventing pipes. This is

exactly the case with cast iron, for here the puffing up due to poor melting practice—and always observed by the metal rising in the gates—means an interior pressure, whereas ordinarily there is a drawing of the metal from the interior to the outside, making for spongy interiors unless feeding with molten metal is resorted to. In the latter case there is no interior pressure, and hence will be no white spots within gray iron, whereas in the case of puffed up metal this condition is quite apt to arise. It may be noted here that these white interiors in the gray fractures of some castings—whereas the natural occurrence would be a white (chilled) exterior and gray interior—are frequently found in stove plate and fittings castings. They are the result of imperfect melting practice, and the foundryman is always anxious to get away from this condition as rapidly as possible.

P. H. GRIFFIN, New York, N. Y. (communication to the Secretary *):—I have noted with particular interest Mr. Sauveur's conclusions on the importance of finding a simple, quick and accurate method of determining combined carbon in cast-iron; also what he says about the importance of the shape and mode of distribution of the graphite particles. For a number of years I have made constant and extensive use of certain physical tests for determining the amount of combined carbon, in connection with other tests for determining other constituents and conditions, viz.: chilling quality and strength of cast-iron. The tests have been carried on as a regular part of extensive manufacturing operations, and have been so thoroughly developed as practical details of the operation that they are not experimental in the slightest degree. The information obtained is direct and positive and permits the regulation of the molten metal as desired with respect to lowering or raising either the combined or graphitic carbon content after the iron is tapped from the cupola furnace. Practice of this kind is carried on to a certain extent in most foundries; but, to be effective, such tests should be made at intervals of from 5 to 10 min. during the entire operation of casting. The regular making, testing, comparing, and recording of such tests on such a scale involves so much detail that it has never been carried out elsewhere, so far as I

* Received Feb. 18, 1913.

know. But, speaking from a definite standpoint established by years of the practice indicated, it can be said that the results are positive and reliable.

Accompanying such tests, at all times, were tests in the laboratory to determine constituents and conditions from that standpoint. But the physical tests gave immediate and reliable information at all times.

The principal tests are six in number, and may be generally explained as follows :

1. *The Combined Carbon or Chilling Effect*.:—Test block 2.5 in. square, 6 in. long, cast in duplicate; one cooled in water as soon as metal solidified, to note condition and show what changes were necessary in the molten metal from which it was taken. Second chill test piece allowed to cool in sand, for examination and filing with daily test records.
2. *Strength and Ductility*.:—Bar 1 in. square, 15 in. long, broken on 12-in. centers to obtain transverse strength; also torsion and deflection in the usual manner. Also for shock tests: weight of 10 lb., dropped 12, 13, 14, and 15 in.
3. *Shrinkage and Fluidity*.:—Strip of metal 12 in. long, 0.25 in. thick on top, 1 in. deep, running to knife edge, cast entirely in sand except ends, which were cast against two iron lugs exactly 12 in. apart.
4. *Abrasion and Hardness*.:—Pencil, 0.5 in. in diameter, cast vertically in chill; bottom of pencil chilled so that crystals of chilled iron stand vertically to chill face.
5. *Structure*.:—Chilled ball, 2 in. in diameter, with plug cast in one side to make small recess, enabling ball to be readily broken in two and thus expose the central shrinkage cavity, which always occurs, and to permit observation of the actual crystalline structure of the mass as shown by the projecting crystals in the shrinkage cavity.
6. *Effect of Constituents*.:—Chilled block, 2.5 in. thick, 6 in. square, cast vertically and cooled in water as soon as metal solidifies.

According to the order of practice, the first three or all of these tests are made at intervals of from 5 to 10 min. throughout each entire day's cast.

In Test No. 1, 1 in. depth of white iron on chill face, with

normal chilling mixture, indicates 0.80 per cent. of combined carbon. (By normal mixture is meant the ordinary mixture used for castings, the quality of which is dependent on the amount of combined or graphitic carbon present, as, for instance, a mixture of the following approximate analysis :

	Per Cent.
Total carbon,	3.65
Silicon,	0.80
Sulphur,	0.12
Manganese,	0.65

In this mixture the sulphur is too high, the manganese also; the former should average 0.08, the latter 0.45 per cent. The constant recasting of scrap with coke in which the sulphur content is always high, and the constant addition of ferro-manganese to neutralize the bad effects of high sulphur, have raised both of these constituents in the castings above normal or proper limits.)

A 1.5 in. depth of white iron in Test No. 1, indicates 0.90 per cent. of combined carbon.

When the test block runs white throughout, the combined carbon indicated is 1 per cent. It may be even higher, as white iron may carry combined carbon up to over 2 per cent. But the point of this test is not so much to establish the actual combined carbon in the chilled test piece as to establish the variations and amount of combined carbon in the molten metal as it runs, and make the test pieces representative of the same.

All the tests are poured at once from a large receiving ladle after each tap from the cupola.

Test No. 2 is the usual test made to determine strength of metal, and when made continuously throughout each cast, becomes a definite basis of comparison.

Test No. 3 is a simple, quick, and accurate means of determining two vital qualities of molten metal.

Test No. 4 determines the resistance of the metal to wear or abrasion. The pencil is placed in a simple apparatus provided with means for grinding the end of the pencil, which has a weight fixed on top to govern contact.

Test No. 5 gives information as to the conformation and shape of the crystals composing the mass. This information is

of the highest value and cannot be readily obtained in any other way.

Test No. 6 is one recently developed and has great possibilities. The cleavage lines brought out in the 6-in. square chilled face, due to sudden cooling in water, can be grouped in distinct classes and varieties, and, while it is too soon to say that the condition and quantity of the different constituents can be quickly determined by this test, all the indications point that way, and, besides, what is most important, the total effect is made plain, no matter what the cause may be. It is a well-known fact that chemical determination of a few constituents utterly fails to account for variations in quality of metal, which may be due to the presence of other constituents not determined.

I have given the details of this practice at some length in order to indicate what has been regularly done in the field.

It is to be hoped that the splendid papers contributed by A. Sauveur and J. E. Johnson, on the subject of cast-iron, will renew the interest of the Institute in this vital subject, and bring out discussion of the subject under the new light that has been cast upon it by the work of later years.

J. E. JOHNSON, JR., Ashland, Wis. (communication to the Secretary *):—I have, first of all, to thank Professor Sauveur for his kind words of appreciation of my paper, *The Effect of High Carbon on the Quality of Charcoal-Iron*, and if I must differ with him in some particulars, it is done in no less kindly spirit than he has displayed towards my efforts.

DISCUSSION OF PROFESSOR SAUVEUR'S PAPER, NOTES ON CAST-IRON.

The Relation Between Analysis and Quality.—A close reading of my paper will show clearly that we realized a distinction between the ordinary analysis and a relatively complete analysis including carbon.

Combined Carbon.—In regard to combined carbon, I wish to say a hearty "Amen" to every word which Professor Sauveur has written. We made the two determinations he describes, total and graphitic carbon, for about a year. But we have abandoned for the present these determinations, for another

* Received Apr. 4, 1913.

reason in addition to the one mentioned by him—the serious errors in the small quantity of combined carbon produced by permissible errors of analysis in the large quantities, whose difference gives its amount. This is, that in pig-iron, and even in castings, the amount of combined carbon varies so greatly from one portion to another that it seemed to us commercially impossible to obtain a sample from the pig which would be truly representative of it. It seems to us, as far as we have gone, that combined carbon is a product of the same conditions as those which control the quality, and is, therefore, an effect rather than a cause. The effect is indicated, for commercial purposes, by our tests for strength, and a more detailed investigation of combined carbon, in view of the uncertainties above mentioned, seemed an unlikely field in which to look for the cause.

We are not fully satisfied in our own minds of the correctness of this point of view, but this, in conjunction with the pressure of other matters, has led us, as above stated, to abandon investigations of combined carbon for the present. If any chemist shall discover an accurate and satisfactory method to determine combined carbon in cast-iron, and also an adequate method of sampling, we shall regard him as a benefactor indeed.

Shape and Size of Graphite Particles.—In regard to Professor Sauveur's remarks under this head, I can only say that if my paper failed to bring out our views as to the importance of this matter, then it has indeed fallen far short of our aim. I had supposed that the importance not only of the shape and size, but of the arrangement of the graphite particles, might almost be said to bristle from every page. If it does not do so, it is merely another illustration of the fact that one can have, in his own mind, so clear a view of the importance of a certain conception that he unconsciously assumes that others will understand it in the same light without special emphasis.

Explanation of White Spot in the Center of Gray Iron.—I think my paper has failed to make clear to Professor Sauveur, in its entirety, my conception of this matter, which may be briefly stated as follows :

1. The higher the carbon in iron the greater the tendency

to form a segregate in the pig in about the same relative position as that found in a steel ingot.

2. This segregate is exposed to three sets of influences: slower cooling, due to the envelope of red-hot iron around it; the graphite-forming tendency of the silicon; and the pressure due to the solidified envelope around the segregate.

In other words, two conditions are necessary for the formation of a white spot:

1. The formation of a segregate due to relatively high carbon.

2. The presence of such an amount of silicon as will make the exterior graphitic without being able to force the graphitization of the segregate against the pressure of the solidified envelope.

This view still seems to me to be correct. Before it can be successfully displaced by another, the latter will have to furnish a valid explanation of the well-established fact that low-carbon irons remain entirely unspotted throughout the whole silicon range, even that which in the higher carbons embraces the spotted iron. This may fairly be called the crux of my whole paper, and if its reasoning is fundamentally incorrect on this point, the value of the paper as a whole fails.

Professor Sauveur's criticism that my microphotograph, Fig. 5, does not represent the eutectic, but what may, for industrial purposes, be called the residual of the eutectic structure, is well founded. This distinction was not sufficiently emphasized in my paper and I am glad to have this deficiency supplied; at the same time, I had thought that my high power microphotographs showed clearly that this structure was one of pearlite and cementite, and that the language used implied that this was indeed the breaking-down product of the eutectic proper.

Carbon Contents of the "Spot."—In regard to the carbon in the segregate and its bearing on the question of whether this is the eutectic or not, there are two important points to be noted.

One is that the white spot in typical spotted iron is more or less intermingled around the edges with gray, and therefore, in drilling a sample for a carbon determination, it is difficult, if not impossible, to avoid getting in some graphite, which may raise the carbon appreciably above that for the white spot

alone. The second is that this white spot, like any white iron, will show both by analysis and under the microscope the presence of a certain amount of graphite. If this graphite be what may roughly be called "primary," as, in my judgment, a part of it is, this will account for a distinct increase in the carbon of the spot above what would be determined by the Wüst and Petersen diagram.

It is well also in this connection to add another correction to that of Professor Sauveur. He says that our irons were not binary alloys of iron and carbon, but ternary alloys of iron, carbon, and silicon. Let us go a step further and say that they are hexanary alloys of iron, carbon, silicon, manganese, phosphorus, and sulphur, and let us remember that at least two of the last three have an influence in keeping the carbon in the combined condition precisely opposite to that of silicon in throwing it into the graphitic condition, and that its departure from the exact ternary ratio on that account not only does not prove that it is not the eutectic, but is, on the contrary, in the direction in which the eutectic for the actual conditions should be expected.

I have no desire to become involved in the controversy raging among metallurgical scientists as to whether the graphite separates out from liquid iron as such, or whether cementite separates out from liquid iron first and graphite is evolved by the breaking-down of the cementite. I cannot understand the close reasoning of the scientific arguments pro and con on this subject. But in regard to the facts I have some information, which, I venture to say, will be corroborated by every furnaceman of experience. Graphite may be seen rising from the surface of the iron as it runs down the runner, sometimes in such quantities as to fill the air with glittering particles like a black snow-storm, and this most frequently occurs when the iron is in its thinnest and most liquid condition. Its temperature I do not know further than that it is several hundred degrees above its freezing point.

If cementite were necessarily formed first, the evolution of graphite in such a quantity would seem to imply the solidification of such a portion of the whole mass as cementite as would materially thicken it and impede its flow. The reverse, as I have stated, actually occurs. Whatever may be the mechanism

by which this transformation is accomplished, the objective facts are as stated.

If, now, the evolution of graphite as such involves the formation of a eutectic of austenite plus graphite, I am most willing to be set straight in this particular. The conception of graphite as being dissolved in the iron in a condition different from that of the other carbon contained therein is difficult for me to form, and I have no opinion to express upon it; nevertheless there is one fact as yet only dimly known and not at all understood, which bears upon it: namely, that white iron is of two kinds, one of which is much softer than the other.

I think it might be well to say that, while my two statements were conflicting from the point of view of the phase rule, as pointed out by Professor Sauveur, the error was a technical one not particularly bearing on the facts or on my explanation of the mechanism of the formation of spotted and high-cleavage irons, because, first, the graphite does separate out copiously from the liquid mass (whether directly or indirectly I do not pretend to say), and this graphite rises towards the top of the pig, or any other mold in which the freezing iron may be contained, whether open or flaked. (In testimony whereof many castings could be displayed whose top surface, to the sorrow of the founder, consisted largely of graphite pockets instead of solid iron.) Second, the eutectic, whether it be austenite-cementite or austenite-graphite, breaks down into a final product of pearlite, graphite, and cementite or ferrite in varying proportions, all of which actions take place in the solid and do not affect the validity of my simple statements.

I have already confessed in the paper itself my inability and even my lack of desire to contribute anything to the advancement of these highly technical questions. Moreover, the paper became far longer and more technical than I desired, in spite of my best efforts to the contrary, and even had I had the power, I should not have discussed these finer points bearing only on the theory of the subject.

After careful examination of Professor Sauveur's acute and penetrating criticism, I cannot escape the conclusion that my facts and such deductions therefrom as are industrially important are left untouched.

For his kind and generous attitude I wish once more to express my thanks.

REPLY TO DISCUSSION OF PAPER,
EFFECT OF HIGH CARBON ON THE QUALITY OF CHARCOAL-IRON.

In reply to Dr. Moldenke's appreciative remarks, it must be reiterated that this paper was not intended as a complete exposition of all the results of our investigation. It was distinctly stated therein that we had certain unconfirmed impressions concerning the subject under investigation, which it would be unwise to publish in the present state of our knowledge. In other words, the paper was not intended to tell all that we thought we had found out, but only to lay a broad foundation for a possible subsequent paper, which should deal with other factors in the problem.

Dr. Moldenke raises the question of oxides in cast-iron. It is probable that we have done more work on this subject in relation to cast-iron than has been done in any other investigation ever made, not improbably more than has been done in all the others put together, but the results are, not sufficiently complete to justify their publication. The investigation is still progressing along these lines, though but slowly, as so many other investigations have to be made at a plant like this, which absorb the time of a limited laboratory force.

I cannot agree with Dr. Moldenke's contention that the grading-card has outlived its usefulness and should be discarded absolutely, and I cannot reconcile this view with his strong agreement with my own conclusions that irons of the same analysis, in the ordinary sense of that term, may differ materially in their physical properties. Dr. Moldenke's confirmation of the latter view carries the authority of his vast experience.

If, now, this point be admitted; if there is a fundamental difference which analysis, as we now know it, cannot find, but which has a profound bearing on the industrial properties of the iron and is reflected in its power to take a chill, why is not the use of the chill itself to determine the extent of these properties perfectly permissible? This is virtually the sole purpose of our grading-card, to collate the results of analysis with chilling power. By these means jointly we can readily separate out the normal irons from those that are abnormally good and those that are below the standard. Even should the

progress of our investigation reveal the ultimate chemical causes for these differences, the convenience and simplicity of this method are so great that we should probably adhere to it for ordinary commercial purposes. To refuse to do so would be to reject the useful fruits of all past experience and of our own investigation.

ALBERT SAUVEUR, Cambridge, Mass. (communication to the Secretary *):—It is very gratifying to me that Professor Howe should so substantially share my views in regard to those characteristics and properties of cast-iron touched upon in my paper.

I am quite prepared to admit that when pro-eutectic cementite is present in cast-iron it can generally be distinguished from the cementite forming at lower temperatures; *i. e.*, from eutectic, pro-eutectoid, and eutectoid cementite.

I am well aware, of course, that, in some cast-irons of suitable composition and after slow cooling, the graphitization of the cementite is complete, such alloys being made up of ferrite and graphite. But these are rather rare instances, and I meant to make clear that the graphitization of cementite proceeds with increasing difficulty and remains more incomplete (1) as the total carbon decreases, (2) as the silicon decreases, and (3) as the speed of solidification and subsequent cooling increases. I also know that graphite may and does form in high-carbon steel, and I have so stated in my paper. Because of their excellence I venture to reproduce here (Figs. 13 and 14) two microphotographs showing this graphite formation in steel, believing that they will be of interest to others. It is natural to suppose that when graphitic (temper) carbon forms in high-carbon steel and in white cast-iron (malleabilizing process) the free cementite graphitizes first, the nuclei of graphite thus formed causing the graphitization of the dissolved cementite.

Certainly when I spoke of iron containing cementite as being in metastable equilibrium and of iron containing graphite and cementite as being in unstable equilibrium, I was using the phraseology of the phase rule. With Professor Howe I realize that this equilibrium, although unstable from the physical chemist's point of view, is, as a matter of fact, remarkably

* Received Apr. 5, 1913.

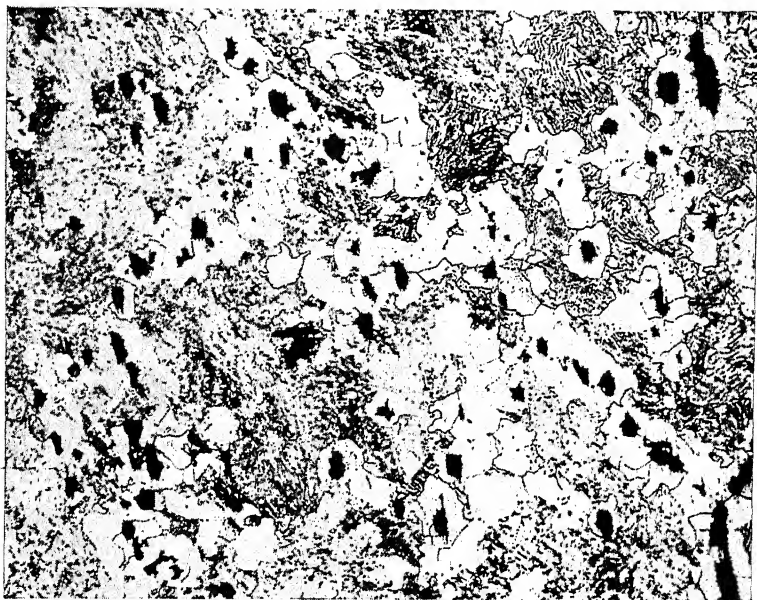


FIG. 13.—GRAPHITE FORMATION IN STEEL.

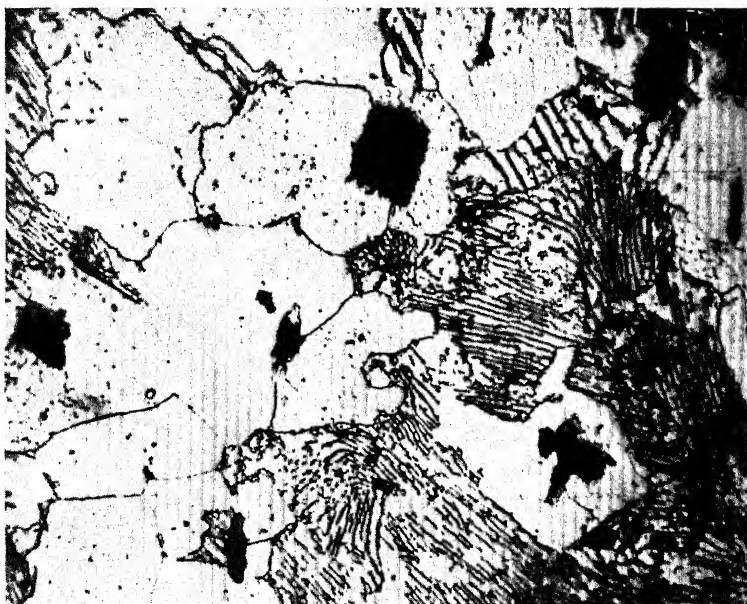


FIG. 14.—GRAPHITE FORMATION IN STEEL.

stable for all practical purposes, and my surprise would have been great indeed had I learned that graphite had formed in the Pillar of Delhi. I firmly believe that although the equilibrium of the metal of that column is metastable, the 0.08 per cent. of carbon it contains will forever remain combined.

Dr. Moldenke has repeatedly told us that cast-iron may and often does contain oxygen, and he sees in the action of this element a possible explanation of the differences in certain physical properties so often exhibited by cast-irons of otherwise identical ultimate composition. This opinion of so eminent an authority should receive serious attention and experiments suggested by it should be undertaken with a view to testing its accuracy. For my part, I feel much as Professor Howe does : *i. e.*, I find it difficult to believe (1) that cast-iron which is produced under such exceedingly reducing conditions can retain an appreciable amount of oxygen, and (2), in the absence of direct and conclusive evidence, that the varying amounts of oxygen it might contain account satisfactorily for many occurrences unexplained by chemical analysis and other tests. Certainly, the burden of the proof rests with those who claim, contrary to what might naturally be expected, that oxygen in appreciable amount is frequently present in cast-iron, and that its influence upon the physical properties is what they describe it to be both in nature and intensity. Dr. Moldenke's statement that cast-iron containing 0.22 per cent. of sulphur machines perfectly in the absence of oxygen, while if oxygen be present 0.10 per cent. of sulphur yields iron of poor quality, is certainly of much importance.

Dr. Moldenke asks how the foundryman is to use a quick and accurate method for the determination of combined carbon, or rather, as I understand it, the results of such method. He asks the question on the ground that the percentage of combined carbon varies from the center to the outside of castings, and that the castings are apt to be made before the metal could be tested for carbon. Surely, if determinations of combined carbon are useless because different portions of the same castings may contain varying percentages of it, then *a fortiori* determinations of silicon must be useless, since these are of value only in so far as they enable us to guess at the probable amount of combined carbon formed under certain cooling con-

ditions. Would it not be of much greater assistance in estimating the grade and quality of cast-iron to be able to ascertain, and to ascertain quickly and accurately, the percentage of combined carbon actually produced under certain conditions, than to be able only to guess at it? Had we a quick and accurate method for the determination of combined carbon, then I would suggest that small bars be cast and drilled under rigidly constant conditions, and that the drillings be analyzed for combined carbon. That would be one way of using the method, and undoubtedly many other useful ways of applying it could be suggested. In regard to the objection that castings would generally be made before they could be tested for combined carbon, it is quite irrelevant. We might as well say that carbon determinations of Bessemer steel are useless since the steel is always made before it can be analyzed.

It is with considerable interest that I note Dr. Moldenke's remarks in regard to the manufacture of castings of great strength through the use of high-silicon metal and hastened cooling. My discussion of such possibility was based upon purely theoretical considerations, and it is a source of much satisfaction to find that they were correct and indeed that they had been anticipated in practice. I wish that Dr. Moldenke would tell us whether those carrying on the operations he described well understood that the increased strength was due in part at least to the presence of smaller graphite particles.

Dr. Moldenke believes that the pressure to which most of us ascribe the formation of "white spots" surrounded by gray metal is due to the occasional presence of dissolved gases or of "oxides in solution." This I am unable to follow. In my opinion, the pressure is caused essentially, if not exclusively, by the contraction of the solidified shell upon the solidifying interior, a pressure which, by preventing the increase of bulk which always accompanies the formation of graphite, makes (under certain conditions) its formation impossible, the carbon remaining combined and the spot "white."

Mr. Griffin's description of the numerous and refined tests conducted by his firm to control and check the quality and properties of their castings is most interesting and will undoubtedly be of much value to others. It is a source of gratification to me that my paper should have been the immediate

cause of the publication of Mr. Griffin's tests. I cannot help the feeling, however, that Mr. Griffin's tests point, and point strongly, to the great need of a satisfactory analytical method for the determination of combined carbon in cast-iron. Indeed, if all or if only some of his tests are necessary because of our ignorance of the amount of combined carbon produced in castings under certain conditions, then our need of an analytical method is urgent indeed.

From Mr. Johnson's remarks it is evident that I should have made it clear that my paper was not a mere discussion of his, but rather a collection of somewhat disconnected thoughts and speculations suggested by his article. Mr. Johnson, for instance, called attention to the importance of the shape and dimension of the graphite particles and his remarks induced me to take up the question also and to add my testimony to his. In doing so it never occurred to me that the reader would infer that it implied a belief on my part that Mr. Johnson had failed to recognize the influence exerted by the size and form of the graphite plates. The evidences I offered were meant to support his contention and not to supply something he had omitted.

Referring to Mr. Johnson's views in regard to the formation of "white spots," I believe that we are in substantial accord. We both hold that these spots are formed in spite of slow cooling and in the presence of silicon because of the pressure exerted upon the interior of the casting by the shrinkage of the solidified and cooling shell. I am not convinced, however, that a "segregate" must form as a necessary condition to the occurrence of a white spot, if by segregate Mr. Johnson means, as I think he does, a portion richer in carbon than the balance of the casting. I venture now the following additional explanation of the formation of white spots, being an attempt to account for the influence exerted by the amount of carbon present.

In hypo-eutectic cast-iron—*i. e.*, in low-carbon iron—the solidification period covers a range of temperature which is the wider the smaller the proportion of carbon, as clearly seen in Fig. 15. Taking, for instance, an iron containing 3 per cent. of carbon, it begins to solidify at L ($1,250^{\circ}$ C.) and continues to solidify as its temperature falls to S ($1,130^{\circ}$ C.). During that period the metal is partly solid and partly liquid—*i. e.*, in

a somewhat mushy condition, if it may be so expressed—and unable, therefore, to exert much pressure on the now solidifying interior, with the result that in such low-carbon iron white spots should not form. If, on the contrary, the cast-iron contains the eutectic proportion of carbon (4.3 per cent. in the case of siliconless iron), it solidifies quickly at *E* (1,130° C.) at a constant temperature and should exert great pressure on the interior of the casting.

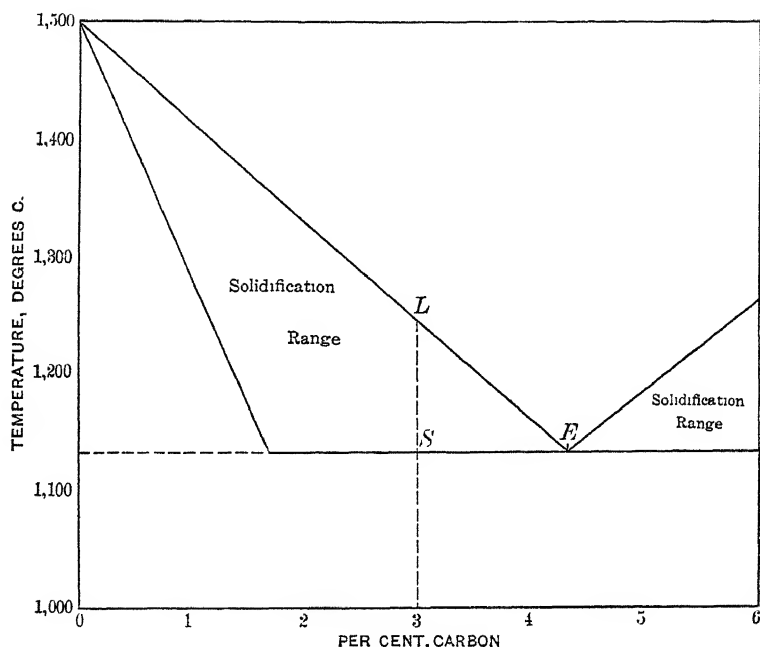


FIG. 15.—SOLIDIFICATION RANGE OF CAST-IRON.

In hyper-eutectic cast-iron the tendency to form white spots should decrease slightly. To sum up, white spots should form the more readily, other factors remaining the same, the nearer the carbon content is to the eutectic ratio. I do not think that the above conclusions are opposed to the results obtained by Mr. Johnson. They seem to me, on the contrary, to be quite in line with them. If my explanation is correct, we should bear in mind the influence of silicon on the proportion of carbon needed to form a eutectic alloy of iron and carbon and remember that it has been shown that the more silicon present the less carbon in the eutectic. It would follow from this that

white spots should form in cast-iron containing relatively little carbon, provided it contained much silicon; but then the direct influence of silicon in promoting graphitization may more than offset its influence in lowering the carbon content of the eutectic.

I suspect Mr. Johnson to have a much firmer grasp and clearer understanding of the equilibrium diagram and of the phase rule than he would have us believe. He has handled both with too much skill and knowledge for us to accept his statement that he has for them the mingled fear and wonder with which the untutored savage regards the pivot-mounted machine gun.

I do not believe that there is at present any controversy raging among metallurgical scientists as to whether cementite or graphite separates out first from liquid form. With very few exceptions, it is now held that cementite is the constituent which falls out of solution when hyper-eutectic cast-iron reaches its freezing point, to break up subsequently into graphite and iron. It is not, of course, a debatable nor a debated question, that if graphite were the constituent to separate from liquid iron a graphite-austenite eutectic would necessarily form at the end of the solidification.

Is it really a fact that the graphite which is seen rising over a thin stream of cast-iron would imply the solidification of a large mass of cementite? Graphite is so light and bulky a substance and its progenitor, cementite, so heavy and compact. Mr. Johnson's statement to the effect that graphite forms while the temperature of the iron is several hundred degrees above its freezing point is somewhat of an anachronism. It is as if he were saying that cast-iron begins to solidify several hundred degrees above its freezing point, for, of course, the formation of graphite, whether directly or indirectly through the decomposition of cementite, means that the solidification point has been reached. Can the formation of graphite from liquid iron be anything else but a solidification process? Is it not the passage of a substance from the liquid to the solid state?

I do not quite understand what Mr. Johnson means when he writes that it is difficult for him to conceive that graphite is dissolved in the iron in a condition different from that of the other carbon contained therein. So far as I know, nobody

believes that carbon is dissolved in iron in two different forms, it being generally held that the whole of that element is in solution as the carbide Fe_3C .

Mr. Johnson's allusion to the existence of two kinds of white cast-iron, one much softer than the other, is rather startling, and it is to be hoped that he will soon give us further information on that point.

Foundry metallurgists and others interested in cast-iron either from a scientific or a practical standpoint will look forward with the greatest interest to the publication of further results obtained by Mr. Johnson in the important investigations he is conducting. If those daily engaged in the production of cast-iron were willing to investigate their problems as Mr. Johnson is doing and with the same degree of skill, intelligence and fairmindedness, the metallurgy of cast-iron would make great strides indeed.

The Production of Solid Steel Ingots.*

BY BENJAMIN TALBOT, MIDDLESBROUGH, ENGLAND.

(New York Meeting, February 1913)

THE problem of segregation and cavities in steel ingots is a subject which has given and is still giving metallurgists, engineers, and operators matter for serious consideration.

This question has come more into prominence lately in the United States, due largely to the desire of railroad engineers to secure a better rail than they have obtained in the past, as the service they demand is increasing in severity; and no doubt manufacturers will be able to meet what is required of them. The rails here are as good as those made in other countries, but the conditions of service and the extremes of climate are more severe and consequently more breakages occur.

Various reasons have been advanced to explain why rails may not be as good in quality to-day as in the past. Some engineers consider that modern methods of manufacturing, designed chiefly to obtain large output, tend to reduce the standard of excellence of more deliberate methods. Others think that 4-ton ingots are worse than those of 2 tons. Again, it is stated that the 100-ton heat in one ladle is too large, and is a step in the wrong direction in casting.

My experience upon the question of the size of the ingot is that in rolling rails of from 85- to 100-lb. sections, the range of ingot is only practically such that the difference in the size does not help in the question of segregation, cavities, or blow-holes.

I call attention to two ingots from work done at Pencoyd, Pa., 16 years ago. These ingots were bottom-poured, cast at the same time from the same center-runner, the one being 20 by 24 in. and the other 13 by 16 in. in size. The steel was acid open-hearth, and contained C, 0.43; P, 0.062; S, 0.069 per cent. At the same place in each ingot where seg-

* Presented also at an informal meeting of the Iron and Steel Committee of the Institute, New York, Nov. 7, 1912.

regation generally concentrates I found C, 0.67; P, 0.13; S, 0.18 in the larger ingot, and C, 0.74; P, 0.16; S, 0.27 in the smaller one. Bottom-pouring at that time was considered better for surface-defects, but the ingots show blow-holes all over the surface, and the smaller ingot, in my opinion, was worse in every respect than its larger neighbor.

The question of 100-ton ladle-heats is an important matter. In my opinion, this is distinctly a step in the wrong direction, as it puts a premium on careless and slovenly casting-work. We find that in order to empty a 100-ton ladle in the necessary time to prevent skulling very large nozzles are used, probably as large as 2.5 in. in diameter. The pressure of this large quantity of steel rapidly enlarges the nozzle, and it would be interesting to know what size the nozzle is when the last portion of the heat is poured. Anyway, large nozzles cause heavy washing up the sides of the molds and this gives surface-defects. There is no doubt smaller ladle-heats, poured with as small a nozzle as the heat will permit, give the most satisfactory results.

We now come to the question of producing sound steel ingots for the heavy trades. There are various methods, but we are forced to rule out the old and well-known Whitworth & Harmer press because of the cost of the process.

Others, among whom is Sir Robert Hadfield, suggest the use of a sink-head to feed the pipe which forms by the shrinkage of metal from its liquid to its solid state. To do this effectively some system of keeping the surface fluid must be adopted. There is no doubt that if this is carried out the sink-head will feed the ingot and the cavity will be in the head. The question to be decided with this method is, whether it is commercially applicable to the large output required from a modern rail-mill.

Sound ingots, as regards the elimination of blow-holes, are produced by means of the well-known powerful deoxidizers, aluminum, silicon, and ferro-titanium. All these deoxidizers have the same effect when used in the necessary varying quantities to produce this. They all produce solid steel except for the large central cavity. They all diminish segregation. In my experience I have found with well-made steel that an addition of 2 oz. of aluminum to the ton is equal to 0.25 per cent.

of added silicon, or an addition of 0.10 per cent. of metallic titanium in the form of ferro-titanium. These additions will all produce the same characteristic central pipe, and if they are used, this piped portion should be discarded in each case. If the rails are milled at each end, which gives a bright surface, as is the practice in England, the pipe is disclosed and the rail rejected by the inspectors. The cost of the aluminum addition is very small; with silicon it is considerable, and with ferro-titanium it is large, to obtain the same result.

If it were not for this large cavity, which may affect as much as 33 per cent. of the ingot, the use of these deoxidizers would improve the quality of the finished rail; but owing to this, they are not used to such an extent as to create this.

It occurred to me that if we were to use a deoxidizer such as aluminum to eliminate blow-holes in the outer envelope of the ingot, and then reduce the area of the ingot, or the top portion, while the center was liquid, the pipe would not form, and that a solid mass would be found in the body of the squeezed ingot.

In analyzing and taking sulphur-prints of the face of a compressed ingot, which was cut longitudinally through its center, I came across an interesting discovery. I found, whenever an ingot was compressed while its center was liquid, that no segregation formed in the center of the upper part, as is usual, but that it was driven to the internal face of the solid envelope in fairly regular percentage over the entire length of the liquid area. The solid outer envelope is the normal steel of the heat and is about 3 to 4 in. in thickness, the carbon in this portion being from 0.65 to 0.70; in the harder portion next this, from 0.75 to 0.80; and in the center about 0.50 per cent. The sulphur and phosphorus also vary in these strata, but as the phosphorus is low in this steel, it was not of sufficient amount to be considered.

In ingots compressed while the center is liquid, without the use of a deoxidizer, I find that the center shrinkage-cavity is not formed, but that the blow-holes, which are found in the outer envelope, are not obliterated and can be traced into the rail. For this reason I prefer to use a deoxidizer, so as not to have any surface blow-holes, which tend to give spongy rails.

The method of procedure is as follows: An ingot of at least

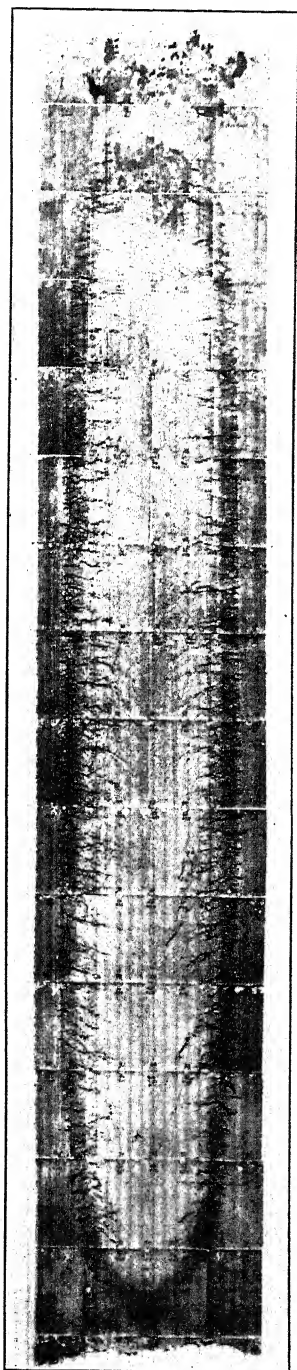


FIG. 1.—SULPHUR-PRINT OF SECTION OF INGOT TREATED BY THE TALBOT PROCESS, SHOWING OUTER SHELL, HARDER RING BACK OF IT, AND SOFTER CENTER.

20 by 24 in. in cross-section is used; 2 oz. of aluminum to the ton of steel is added to the ingot as it is being poured. Aluminum is preferred because of its low cost and its low melting-point. It causes a perfectly solid outer envelope to be formed, and solidifies the metal earlier than if no deoxidizer were used. The ingot therefore can be stripped earlier, and it is then put into the soaking-pit to allow the envelope to become thicker, and at the same time have a proper temperature upon its surface for compression. A 20- by 24-in. ingot is reduced to about 18 by 18 in., and it is then returned to the soaking-pit for a proper heating and solidifying of the mass. After this has

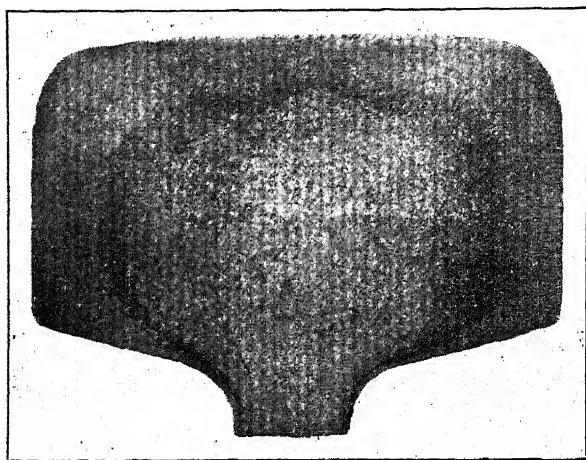


FIG. 2.—RAIL-HEAD TREATED BY TALBOT COMPRESSION METHOD.

The analyses of the outer area, the dark ring, and the center are as follows :

	Outside.	Ring.	Center.
	Per Cent.	Per Cent.	Per Cent.
Carbon.....	0.59	0.65	0.42
Sulphur.....	0.055	0.076	0.038
Phosphorus.....	0.027	0.028	0.018
Manganese.....	0.73	0.74	0.69

been accomplished, it is rolled down into a bloom, cropped, and passed to the rail-mill. The rail produced has the same characteristic formation as the squeezed ingot; viz., a hard working-face, a harder ring at back of this, and a softer center.

It is the question of this new formation that we manufacturers have to discuss with railroad engineers and metallurgists.

If they accept this new structure with the guarantee that in this formation they have no pipe in a rail, then it will be for manufacturers to consider laying down the necessary preparatory plant to accomplish the liquid compression of the ingot, as it cannot be properly accomplished in any existing rail-mill without largely decreasing the output.

So far the rails produced by this method have been tested under the drop to the British standard specifications and they pass this test satisfactorily.

The question of taking tensile tests out of the head has still to be considered, as these will vary as they do to-day according to the position selected. In fact, small tensile tests are unsatisfactory at the best, and it would appear that the only satisfactory method would be to have the testing-machine large enough to pull the full-sized head.

The sulphur-print in Fig. 1 shows the throwing of the segregates towards the inner wall of the solid envelope. The black line contains higher carbon, higher sulphur, and higher phosphorus than the outer envelope or the purer center. When rolled to the rail the head will show the normal analysis, there will be a ring of harder metal underneath it and softer metal in the center of the rail, Fig. 2. The formation is the same whether or not a deoxidizer is used excepting that you can trace the blow-holes in the outer envelope where a deoxidizer is not used, and this, of course, is not desired.

It must be distinctly understood that this sulphur-print does not accurately represent the percentages of sulphur present, as these sulphur-prints make the sulphur look higher than it really is, and it should be noted also that the horizontal lines are not fissures. For this reason, I have decided not to use the sulphur-prints for publication on the rails, as they are misleading. The illustration of the etched rail-head shows the formation better, and it is in this form I hope to publish the sections of the rails.

It is also worth noting that, in the section of the etched rail-head, the outer envelope has the normal analysis of the heat, that the ring contains higher carbon, sulphur, and phosphorus, and that the lighter center is purer than the rest of the section. This remark also applies to the sulphur-print of the pressed bloom.

DISCUSSION.

PROF. HENRY M. HOWE, New York, N. Y.:—It seems as though Mr. Talbot has taken the rational way to close up the pipe. He has closed up the pipe by the cheapest possible means, running it through a rolling-mill.

H. D. HIBBARD, Plainfield, N. J.:—I should like to ask how many passes it takes to effect the reduction of ingots from 20 by 24 in. to 18 by 18 in.

MR. TALBOT:—Very few; I cannot say just how many.

MR. HIBBARD:—The general idea, as very likely Mr. Talbot knows, is very old. It was long ago tried, or at least suggested, to roll ingots while the center was liquid. I have here two patents, one to Leighton, the claim of whose patent is:

“The herein-described method of rendering cast-steel ingots solid which consists in rolling the ingot while its interior is still in a plastic condition, substantially as set forth.” I think that was tried by the Pennsylvania Steel Co., and I take it it is the underlying idea here.

MR. TALBOT:—You must put the ingot back in the pit to get it in condition: what you refer to is simply “rolling green steel,” which nobody wants to do. When that occurs, we stop the mill.

MR. HIBBARD:—The other patent came out in 1898. In one case the rolls are bulged and in the other the ingot is bulged so that the cavity will be filled when the ingot is rolled.

I remember hearing some years ago that rolling an ingot from its initial heat was tried in Pittsburg and the result was a squirt of steel all over the shop.

MR. TALBOT:—Rolling green steel cannot be done. When it comes from the rolls it has fins, which produce serious defects, and the fins must be rolled in. In addition, the ingot is likely to tear.

WILLIAM R. WEBSTER, Philadelphia, Pa.:—An examination of the Watertown series of tests shows that this has been done and that the interior of the ingots was fractured.

A. A. STEVENSON, Philadelphia, Pa.:—I have not had much experience as to the effect of working green steel. Most of my experience has been confined to the working of steel which has been allowed to become cold in the ingot form. Certain tests that were made at Watertown Arsenal several years ago show the danger of transverse cracks in the bloom where the steel is rolled in the green state. Some of those present were on the committee that had the tests in hand and undoubtedly will know to what I refer. Some years ago we had trouble with blooms which we purchased for forging purposes. The results of investigations at the manufacturer's works showed that the trouble arose from rolling green steel, and when this practice was abandoned the trouble ceased.

E. F. KENNEY, Johnstown, Pa.:—I should like Mr. Talbot to tell us something about the time-elements. We tried it according to the data given in Stead's statement and we did not get good results. It seems we did not get the right length of time because we did get cavities and we did not get this soft center. Can you tell us about that?

MR. TALBOT:—What size ingot did you take?

MR. KENNEY:—19 by 23 in.

MR. TALBOT:—Did you use a deoxidizer?

MR. KENNEY:—Yes: 4 oz. of aluminum to a 3-ton ingot.

MR. TALBOT:—Not quite 2 oz. to the ton.

MR. KENNEY:—We stripped it 27 min. after teeming; left it in the soaking-pit for 42 min. and then rolled it down to 18 by 18 in.

MR. TALBOT:—Did you cut it open?

MR. KENNEY:—Not then. We subsequently rolled it down to 9.5 by 9.5 in. and in this found the cavities, and an analysis did not show the soft center. We rolled a second ingot, but we did not stop until we got it down to 12 by 12 in. and then it split and the liquid center ran out.

MR. TALBOT:—You were evidently too late on your first ingot. The metal had become too solidified. Or possibly you simply let it bulge, and so did not reduce the area sufficiently.

MR. KENNEY:—Yet we had liquid metal in the interior, for it ran out when the rolling of the second ingot was continued to 12 by 12 in.

P. H. DUDLEY, New York, N. Y.:—I have been much interested in Mr. Talbot's description and am as much puzzled as Professor Howe, as to how Mr. Talbot is able to get a center of the ingot with so much less carbon than the outside. Mr. Talbot sent me from England a copy of his paper, and I replied that it was surprising to me how he obtained so soft a center in the ingot. I do not understand that point yet. It has not been sufficiently explained.

As to what Mr. Talbot has said of the use of deoxidizers to make the outside of the ingot solid, I have been using ferro-titanium and silicon for many years to make sound steel and free from blow-holes. We charge the ingots into the reheating-furnaces very soon after being stripped, and do not allow the full shrinkage-cavity to form, due to the cooling of the ingot from hot to cold metal, and there is scarcely a trace of the cavity shown in the crop of the rolled bloom. We have been doing this with different size ingots very successfully for several years. To date we have had the rails made from large basic open-hearth ingots in service only about four years, but pipes have not developed in the webs, and but few split heads have occurred. The illustrations of ingots which I have in my hand are those shown in my discussion of Professor Howe's paper.¹ These show that we obtain sound ingots free from exterior blow-holes, though, of course, we have to discard enough to cut off the upper unsound portion, which is from 9 to 12 per cent. in the different sized ingots, but when they are promptly charged into the reheating-furnaces, they do not show cavities in the cut rail-blooms.

When we use a subsidiary deoxidizer with the larger ingots, we may find a trace of cavities in the crops, because we are unable to handle the ingots with sufficient rapidity to prevent their formation.

¹ *Trans*, xl., 825, 826 (1909)

We made a 19 per cent. discard for special work in 19- by 19-in. ingots, and in part of the heats we used twice as much extra deoxidizer as in the others, and as we are not able to change the rapidity of the mill-practice, a small cavity formed, as would be expected, though it was all removed in the discard.

MR. WEBSTER:—You do not put the ingots back in the furnace before rolling?

DR. DUDLEY:—No.

MR. WEBSTER:—Then you do what we call rolling green steel.

DR. DUDLEY:—We do not roll it so quickly as to eject any steel; it is pasty, but not green steel. By our mill-practice, we have nearly eliminated piped rails, and have much better metal, which is giving us excellent service. This is by no means new, as I used the same mill-practice in 1890, when I introduced the low 0.06 phosphorus and from 0.60 to 0.65 carbon rails for the Boston & Albany, Mohawk & Malone, and later for the New York Central & Hudson River Railroad Company.

The small ingots for two or three 100-lb. 30-ft. rails, until 1898, were charged into a horizontal reheating-furnace, and we did not have piped rails nor roll green steel. The deoxidizers in the chemical composition were so arranged as to make sound steel, which was, as already described, handled so quickly after stripping that there was hardly a trace of a shrinkage-cavity in the discard of the crop from the bloom.

W. C. CUSHING, Pittsburg, Pa.:—I was very much surprised to notice that there is so much segregation in the interior. I do not feel at all certain as to what would be the action in service of a rail with a hard exterior and a quite soft interior. It would be interesting to follow up that point if we had an opportunity to try in service some rails made in accordance with this method. It would be mere speculation as to what would happen. We look at the present time on rails such as are shown in the illustrations with considerable suspicion, as we find that the etchings of our worst cases of failure show rails of that type. At present the segregates are usually higher in the center, but how it would be in the reverse

case, with negative segregation, I am unable to say. I would be very much interested in seeing the result from experience. We are watching with great interest the plans for production of sound ingots, as we feel it would remove from the field of discussion the question whether we should have a definite discard from ingots or not. The nearer sound ingots are produced the less that question will come before us. I am therefore greatly interested in having these various methods which were proposed at the Iron and Steel Institute—Mr. Talbot's, Sir Robert Hadfield's, and Dr. Goldschmidt's methods—tried out, so that we may know what may be expected. I very much hope the steel companies will find it to their advantage to prosecute the investigations further and keep us posted.

M. H. WICKHORST, Chicago, Ill.:—I am something like Mr. Cushing. I came to sit at the feet of the master and learn from him. I have been very much interested and consider the information that Mr. Talbot has given us to be of a good deal of value. I may perhaps talk a little in reference to the matter that Dr. Howe brought up. Our survey of ingots has shown that there is generally some negative segregation in the interior and lower half of the ingot; that is, the metalloids, carbon, phosphorus, and sulphur, are below the average of the steel as poured from the ladle. The well-recognized positive segregation or concentration of these elements occurs in the interior and upper part of the ingot. There is also more or less tendency of the upper and exterior portions of the ingot to show negative segregation. When sufficient deoxidizer has been used, such as silicon or titanium, we do not find the usual spongy or porous condition of the metal in the upper part of the ingot, but when cast in ordinary open-top molds in the usual manner the interior shows a large central cavity or pipe. Such metal shows the negative segregation in the interior and lower part of the ingot, and some positive segregation around the pipe, but not concentrated to the same extent, due perhaps to the metalloids not being able to concentrate into so small a volume on account of the presence of the large pipe; that is, instead of being concentrated close to the axis, they are distributed around the pipe something like a ring. Such

deoxidized metal also seems to show less negative segregation in the upper corners of the ingot.

In this connection I present as interesting a diagram,² Fig. 3, showing the distribution of phosphorus in a large Bessemer ingot about 25 by 30 in. at the bottom and 58.5 in. high, that weighed 10,600 lb. The average composition of the steel was: carbon, 0.54; phosphorus, 0.093; sulphur, 0.035; manganese, 0.99; silicon, 0.13. The phosphorus-content in the various areas was as follows: *A*, below 0.070; *B*, from 0.070 to 0.085;

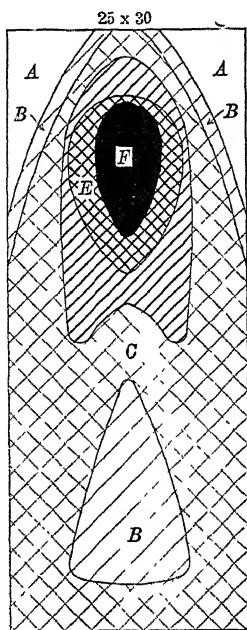


FIG. 3.—DISTRIBUTION OF PHOSPHORUS IN A BESSEMER STEEL INGOT.

C, from 0.085 to 0.105; *D*, from 0.105 to 0.135; *E*, from 0.135 to 0.175; *F*, above 0.175. With smaller ingots the separation was not as marked, however, as in the one shown.

GEORGE L. FOWLER, New York, N. Y.:—I should like to ask if there is any possibility that the churning action due to the rolling of the ingot has any effect which produces the soft interior. It must be mechanical or it would occur in other ingots.

² *Proceedings of the American Railway Engineering Association*, vol. xiii., p. 689 (1912).

MR. TALBOT.—I am afraid I cannot answer. I do not know why it happens, but it happens. I have been trying to find out from the professors, but have not obtained very satisfactory answers.

BRADLEY STOUGHTON, New York, N. Y.:—In order to put a little different view on this question, I would like to make a few suggestions, particularly to those present who are familiar with the structure of bled ingots. We know that crystals grow out from the surface of an ingot towards the interior. That such a condition does exist is demonstrated by the copper-tests of Sir Robert Hadfield. With the crystals growing out—and growing out further every minute—if you put pressure on the outside of the ingot the crystals are going to meet in the center, and this purer metal, so to speak, displaces the liquid metal and forces the segregate out nearer the edge. Perhaps from this viewpoint some one can give us a better explanation.

WILLIAM R. WALKER, New York, N. Y.:—I should like to ask if this peculiarity of structure takes place with great regularity.

MR. TALBOT:—We have not yet made a great many.

MR. WALKER:—Do your own tests show the ingots regular?

MR. TALBOT:—Yes.

A. A. STEVENSON:—I would like to ask as to the percentage of discard Mr. Talbot found practicable with his method.

MR. TALBOT:—We simply cropped off a few inches.

MR. STEVENSON:—We have heard quite a little about negative and positive segregation, and my next question may indicate negative knowledge and positive ignorance. I should like to ask whether there might not be some danger that the rolling action of the wheels passing over the rails might not cause a separation between the outer envelope and the highly segregated shell, or between the segregated shell or ring and the soft interior. Some years ago one of our leading roads made experiments with Harveyized tires. As I remember the result of the experiments, the mileages obtained promised to be good, but after the tires had been in service a short time they

gradually shelled out; the separation occurring between the hardened surface and the softer interior. Some of the railroad people present may be able from their greater experience to enlighten us upon this point.

MR. WEBSTER :—I desire to call attention to Captain Hunt's remarks about the hard exterior and soft center which causes splits. Is there not a great difference between the layers of steel here and those referred to by Mr. Stevenson? Is not the union entirely different from that Mr. Stevenson had in his tires?

PROFESSOR HOWE :—Is that hard carbon ring of such size as to act as an anvil against which the other parts are pressed in the rolling of the rail?

MR. TALBOT :—I do not think so. We speak of it as being much harder, but there is 0.56 carbon in the outside and 0.77 in the ring, and then the center comes down to between 0.45 and 0.50. It is not a soft-steel center, but merely softer steel.

CHARLES KIRCHHOFF, New York, N. Y. :—Have the drop-tests carried out shown in any way any separation of the harder exterior and the softer center?

MR. TALBOT :—No. We are relying upon the drop-test. We drop rails all the time. If it would not stand the drop it, of course, would not be commercial. But they are perfectly satisfactory.

There has been a new term used here to-night; I should like to know what is "negative segregation?"

PROFESSOR HOWE :—Segregation is an enrichment, and negative segregation would therefore be an impoverishment. You cannot have an enrichment in one place without impoverishment in another.

MR. TALBOT :—It is possible that these pictures or sulphur-prints may have frightened you unnecessarily. They may show the differences in greater contrast than they are practically. In the treatment of the prints with the acid the situation may be made to appear worse than it really is; the analy-

ses are the real proof. The drop-test is what we have relied upon.

I intend to continue until I absolutely satisfy myself that this is going to be perfectly regular, as I believe it is now. Of course we have not made thousands of tons, as we shall do when we have a plant, as I believe it is a step in the right direction. In England some engineers require pulling-tests, but most of them rely on the drop-test. They say that when I am satisfied that we can produce solid ingots commercially they will reduce the guarantee conditions as to cropping which we give them.

MR. WEBSTER:—Have they not said to 5 per cent.?

MR. TALBOT:—Yes.

A VOICE:—Will you kindly describe how you make these sulphur-prints?

MR. TALBOT:—It is very simple. In England the paper used is called "Vigorous Glossy Velox." It is prepared by coating with an emulsion of gelatine and silver bromide, and is soaked in a 2 per cent. solution of sulphuric acid. The prepared surface of the paper is placed on the bright cleaned surface of the ingot or rail to be printed. The air is carefully squeezed out so as to get the paper into close contact with the steel surface, and it is left for 1 min. It is then taken off and washed in water, fixed in sodium hyposulphite, again washed, and then dried. The print shows the presence of sulphur in innumerable black dots, due to the action of hydrogen sulphide gas liberated from the manganese sulphide in the steel by the action of the acid, the gas forming a black compound of silver sulphide, which is permanently fixed on the paper. If a strong solution is used, a darker print is obtained.

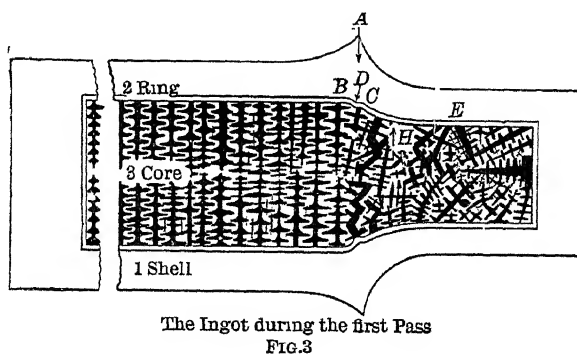
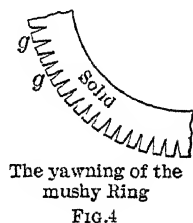
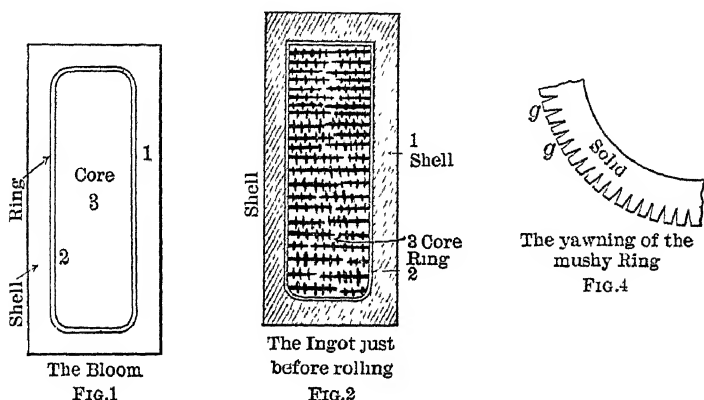
HENRY M. HOWE, New York, N. Y. (communication to the Secretary *):—Why is Mr. Talbot's ring enriched at the expense of his core?

In the blooms and rails which result from Mr. Talbot's process there are three well-marked layers, the shell, 1, in Fig.

* Received Nov. 11, 1912.

1, the ring, 2, and the core, 3. The shell is of about the normal ladle composition. The ring is greatly enriched in the three segregating elements, carbon, phosphorus, and sulphur, and the core is correspondingly impoverished in them. Manifestly, something has happened to transfer these segregating elements from the core to the ring. What is it?

If we put together the remarks made by several gentlemen, especially Professor Stoughton and Mr. Kenney, and supply some missing links, I think we have a fair explanation.



First we must recognize that, because of the thorough deoxidation which Mr. Talbot uses, there would be very little segregation if the ingot were left undisturbed, and solidified in the normal way without compression by Mr. Talbot's process or in any other. In other words, the solidification is of that exaggerated landlocking type, which was noticed in the early use of aluminum. The metal produces the impression of solidifying

suddenly from shell to core, which means that, because of the stillness caused by the suppression of the gas convection currents by the deoxidizer, pine-tree crystals shoot out into the molten mass very far in advance of the actual completion of solidification. Fig. 2 may show the condition of affairs diagrammatically after solidification has progressed a certain distance. Here we may recognize three tolerably well defined concentric layers, 1, that part of the crust in which solidification has gone so far that the metal is strongly coherent and does not fissure when elongated by the forward pull of the rolls; 2, the ring in which the metal is in the mushy stage between grass and hay, when it has neither the cohesion of the solid layer 1 nor the mobility of the core; and 3, the core, which is a veritable sponge, composed of dendrites of purer material interpenetrating the still molten and less pure metal. This is the condition of things when the first pass through the rolls occurs, and because of this arrangement that pass and the succeeding ones transfer this still molten and less pure metal out from between the dendrite trunks and branches to the ring, as I will now try to explain.

As any given transverse section of the ingot enters the rolls, the first stage of the effect caused in it by the rolls is shown at *A* in Fig. 3. This is in effect a powerful compression. On the inner face of the shell the metal at *C* is forced backwards towards that at *B*, and this compression would naturally squeeze in the direction of the arrow *D* part of the less pure molten metal locked up between the purer and more solid parts of this layer. Thus comes about a first enrichment of the ring, and of that layer of the spongy core in immediate contact with that ring. Impure, less firm metal is forced from the inner part of the shell into the ring, and from the ring into the adjacent part of the core.

In the second stage of the action of the rolls, from *C* to *E*, the shell is elongating. But the metal in the ring will not elongate because it is in the mushy incoherent stage in which any attempt at elongation simply tears it open. Because the shell is elongating rapidly while the ring is not, the metal in the ring tends to gape open in minute cracks as sketched diagrammatically in Fig. 4.

But while the ring is thus cracking open in thousands of

minute cracks, it is pressed strongly down upon the spongy core.

If we press a glass plate down upon a sponge filled with ink, we drive that ink out of the pores of the sponge into the relatively open space between the sponge and the glass plate. If we have a long cylindrical ink-wetted sponge which roughly fills the interior of a test-tube, and if we then press down on top of that sponge with a plug that fills the bore of the test-tube closely, so that the ink cannot rise past it freely, in like manner we squeeze the ink from the pores of the sponge into the space between the sponge and the glass. In like manner between the solid walls, 1, of our ingot there lies a sponge, consisting of purer dendrites between the boughs of which is the residual still molten impure metal, that which when later revealed by the sulphur-print blackens like ink. The radial pressure of the rolls crushes this sponge down into a mass of distorted trunks and branches, bent back and doubled on each other, and expels from its pores and towards the surface of that sponge the molten impure metal. This outward movement of this impure molten metal is greatly exaggerated by the fact that at this very instant the little cavities shown at *g, g, g* in Fig. 4 are just opening, so that it is as if, at the instant when we press down on the stopper in our test-tube, we simultaneously converted its inner face into a bibulous sheet of blotting-paper, full of fast-growing pores, which suck up the outward squeezing ink.

Thus the tearing open of numberless little cavities in the inner face of the ring at this instant, because that incoherent face cannot elongate with the shell to which it is integrally attached, furnishes a haven for the outward-pressing impure molten metal which the radial pressure of the rolls leads to travel in the direction of the arrow *H*. That pressure forces the relatively coherent immobile dendritic branches centripetally, and because of that centripetal movement of the less mobile branches, the more mobile impure molten between them migrates centrifugally as shown by the arrow *H*.

The entrance of these centrifugally traveling molten particles from the outer part of the spongy core into the yawning cavities in the ring, makes way for the like centrifugal travel of the deeper-seated particles of the molten, lying nearer the axis

of the ingot. At the next pass the ring itself is torn open in like fashion, and into these new cavities as they open, the deeper-seated parts of the impure molten are now squeezed, so that each squeeze of the rolls carries farther this squeezing of the impure molten from deeper and deeper-seated layers of the core into the recurring cavities in the ring.

Thus is the ring enriched in the segregating elements at the expense of the core, which is correspondingly impoverished or purified. If this is not the whole story, it seems to be a large part of it.

In reply to a question from Mr. Talbot, "segregation" is etymologically the "flocking apart" of the three segregating elements, carbon, phosphorus, and silicon. It implies an enrichment of the region into which they flock; but this enrichment of one part of course implies a corresponding impoverishment or purification of the part whence these impurities segregate.

The expression "negative segregation" seems unfortunate and misleading. Is it not better to call it just what it is, "an impoverishment"? The act of segregation enriches one part, called the "segregate," at the expense of others, the impoverished parts. If a specific name corresponding to "segregate" is needed for the parts thus deserted, how would "the desert," *i. e.* the part deserted, do?

MR. HIBBARD (communication to the Secretary *):—In considering pipe in ingot-steel and its cure one should keep in mind its method of formation. The causes which produce pipe begin to act as soon as the ingot is teemed and continue until solidification is complete. The causes being continuous, it seems reasonable to hold that the cure or preventive means should likewise be continuous, and experience seems to bear out the correctness of this view. There is no one instant at which a cure may be applied which will be wholly effective. Further, it should be remembered that steel pipes from physical rather than from chemical causes; that is, from the contraction of the metal as it loses heat, and also because the outside of the ingot is cooler than the inside. Therefore the cure should be physical or mechanical. If the ingot lost heat at a uniform rate throughout, the pipe would undoubtedly be smaller,

* Received Nov. 29, 1912.

if not wholly absent. Many plans proposed for doing away with pipe have been futile because they were not continuous acting.

In a certain sense Mr. Talbot's plan is continuous because the several passes through the rolls may each be curative of the pipe already formed up to that time. There is likely to be some tendency to form pipe, however, after the last of the preliminary passes when, as apparently must be the case sometimes, a part of the interior is still liquid. The plan may do well in actual work, nevertheless, as things sometimes do which are not ideal or are even unmechanical.

The ideal treatment of pipe is perhaps to be met in Harmet's plan, but he goes too far and presses or draws the solid parts of the ingot as well as the piping part; all very nice, but greatly adding to the expense of carrying out the process because of the very powerful apparatus required.

The center, purer than the ingot average, which results from Talbot's treatment, and Stead's explanation thereof, gives quite a new view of segregation. The mother-liquor, containing more than the average contents of the various steel-ingredients, is squeezed out from the purer solid or near-solid material, as water is squeezed from a sponge, and the purer center results. This shows that segregation is really a worse thing than has been heretofore usually believed. Drillings from the segregated part of an ingot are probably always diluted by some of the purer metal intermixed, so that their analysis does not tell how impure the true segregate really is.

MR. TALBOT (communication to the Secretary*):—In the discussion on my paper very great surprise was shown at the fact that in the sulphur-prints exhibited, there was a regular area of higher carbon and sulphur metal upon the inner wall of the solid envelope, and a purer center inside that. The question was raised whether this formation was regular when the ingots were treated uniformly, and doubts were expressed of its being consistently found. Further experiments have shown that it can be consistently relied upon, and there is no freak in the formation of this structure. The photograph of the head of a treated rail, Fig. 2, depicts this structure more

* Received Feb. 8, 1913.

correctly than did the crude sulphur-prints exhibited in New York, which exaggerated the structure, and might, therefore, cause unnecessary criticism from metallurgical railway engineers. In fact, one of the members looking at the sulphur-prints thought that there might be a distinct cleavage in the metal, whereas, there is nothing of the kind, there being a gradual gradation of one structure into the other.

The results in Table I. are of Brinell tests made on the identical rail-head of which Fig. 2 is the photograph.

The sulphur-print, Fig. 1, giving a longitudinal view of a squeezed ingot cut down the center, shows the structure developed by the Talbot process. Drop tests have been made on many rails treated by my process, as reliance is placed chiefly on this test to see if there is any irregularity; and we find the results very satisfactory, as shown in Table II. A series of tensile strength tests is given in Table III.

Assuming that a sound rail with this structure is accepted by rail-users, we have then to consider the practical adoption of the process in steel-works. There is no doubt that for open-hearth steel rails, the large 250-ton tilting furnaces, giving heats of from 50 to 60 tons, irrespective of how the furnaces are operated, can give the steel in a more regular sequence, and are under better control as regards time of tapping, than the so-called 60-ton fixed furnaces.

As to the casting department, I may say that, in my opinion, the present ordinary methods have to be revised, and the casting must become an important department of steel manufacture. To-day, both metallurgists and works operators have neglected this department, and have not considered it as of anything like the same importance as the actual manufacture of the pig-iron, or the conversion of the iron into steel.

There is no doubt that the railway engineers have hit upon that point in the manufacture of steel rails which is causing most defects, and if given sound ingots the troubles now encountered in rail manufacture will be avoided. To my mind the casting-department should have the attention of men skilled in the art, to remove the slur now put upon steel-makers in reference to cavities and blow-holes, which form in the ingot during the solidification of the steel in the mold.

I fully realize that, if liquid compression of the ingot is to be adopted, steel furnaces must not be allowed to "bunch" together and to tap a large number of heats at once, as is the present practice. If a continuous set of rolls be used for liquid compression, then such rolls must be equal to dealing with the maximum quantity of steel tapped in any given time; but, speaking as a works engineer and practical operator, I see no difficulty in designing a plant to squeeze the required number of ingots to keep a modern rail-mill supplied.

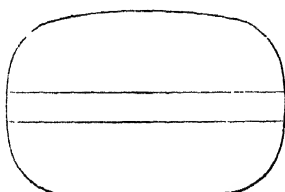
Assuming that the necessary preparatory treatment has been made, we find that the size of the ingot must be standardized when such treatment is used, and a 4-ton ingot seems to be a most useful size in this respect.

Finally, in my opinion, the characteristic structure developed in rails treated by my squeezing process must cause these rails to be better than rails from the upper portion of ingots from ordinary heats, which are always more or less segregated in the center.

I hope to give full details of the results we have obtained in a paper to be read before the Iron and Steel Institute in London next May. I was under promise to give this paper before I visited America last year.

TABLE I.—*Brinell Hardness Tests Across Head of Talbot Steel Rail.*

Longitudinal Horizontal Section through Rail-Head.



Width	Brinell Hardness, Number..... ..	228	50.25 tons per sq. in.
	Brinell Hardness, Number..... ..	228	50.25 tons per sq. in.
	Brinell Hardness, Number..... ..	228	50.25 tons per sq. in.
	Brinell Hardness, Number..	202	46.00 tons per sq. in.
	Brinell Hardness, Number..	217	49.47 tons per sq. in.
	Brinell Hardness, Number..	228	50.25 tons per sq. in.
	Brinell Hardness, Number... ..	228	50.25 tons per sq. in.
	Brinell Hardness, Number... ..	228	50.25 tons per sq. in.
	Brinell Hardness, Number... ..	217	49.47 tons per sq. in.

TABLE II.—*Drop Tests on Talbot Steel Rail.*

CARGO FLEET IRON COMPANY, LIMITED.

FALLING WEIGHT TEST.

Made on 88½ BH., E. I. R. Rails.

Weight of Tup, 1 ton. Length of Rail Tested, 5 ft. 0 in. Bearings, 3 ft. 6 in. apart.
W. 13900.

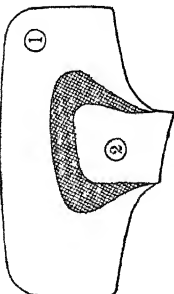
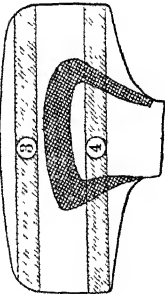
Date	Cast No.	No. of Test.	Deflections						Remarks.
			1st Fall. 7 ft.	2d Fall. 20 ft.	3d Fall. ft.	4th Fall. ft.	5th Fall. ft.	6th Fall. ft.	
31-1-1913	69	1	1.08	3.70	Bull headed rail, 88½ lb. per yard, British standard specification.
	69	2	1.05	3.70	
	69	3	1.12	3.85	
	69	4	1.18	3.90	
	69	5	1.15	3.90	
	69	6	1.18	3.95	

TABLE III.—*Tensile Strength Tests on Talbot Steel Rail.*

CARGO FLEET IRON COMPANY, LIMITED.

TENSILE TESTS.

W. 10,367—13-5-12—2,500.

Description. Ingot No. 69	Date.	Blow.	No. of Test.	Dimensions of Section of Test Piece.			Breaking.		Elongation		Dimensions of Section after being broken			Reduction. Per Cent.	Elastic Limit.		Remarks.
				Breadth	Thick.	Area.	Strain on Section in Tons	Strain sq. inch in Tons.	Total. 3/4	Per Cent. 1/2	Breadth.	Thick.	Area.		Total in Tons.	Per sq. inch in Tons.	
	69	1	1	Di- ameter	0.58	0.204	13.1	49.6	13	15	Di- ameter	0.515	0.208	21.2	60 1/2 Granular, 40 1/2 Fibrous
	69	2	2	Di- ameter	0.57	0.255	11.6	45.5	14	17	Di- ameter	0.51	0.204	20.0	60 1/2 Granular, 40 1/2 Fibrous
	69	3	3	1.0	0.22	0.220	10.9	49.5	13	15	0.91	0.195	0.177	19.6	70 1/2 Granular, 30 1/2 Fibrous
	69	4	4	1.0	0.76	0.260	12.5	48.1	11.5	14.5	0.925	0.235	0.217	16.5	85 1/2 Granular, 15 1/2 Fibrous

Piping and Segregation of Ingots of Steel and Ductility-Tests for Open-Hearth Steel Rails.

BY P. H. DUDLEY, NEW YORK, N. Y.

(New York Meeting, February, 1913.)

THE piping and segregation of ingots of steel is a comprehensive subject, and must be studied in reference to the different grades of soft, medium, and hard steel, rather than to discuss it in a general manner, as though applicable to all grades. Rail steel, however, will receive the most consideration in this paper, for the wheel-loads of the motive power and rolling stock have doubled from 1890 to 1910. Steel rails and tires which were efficient under the light loads, became less serviceable under the double wheel-loads, quadrupled tonnage, and higher speeds.

Bessemer Steel.

Bessemer steel of from 0.10 to 0.15 per cent. of carbon, for splice-bars, spikes, and tie-plates, rises in setting, and is cast in bottle-mouthed molds, which must be capped to prevent an overflow from the top. This grade of steel rises in the molds and makes a longer ingot than the volume of molten steel when first teemed. The ingots which are allowed to cool and then cut open, show, particularly in the upper part, large occluded blow-holes, and when they are not oxidized or do not contain foreign matter, weld more or less completely when the steel is rolled or forged above $1,100^{\circ}$ C., and it is in this way that the blow-holes are closed in the low-carbon steels.

Boiler-Plate and Fire-Box Steel.

These grades often contain more or less minute laminations, which are the remains of small blow-holes forming after the setting metal has reached a pasty condition. The blow-holes in the low-carbon steels have not been prevented by using deoxidizers, though the ingots are slightly improved so far as the soundness of the steel is concerned. This grade of steel also rises in the molds in setting.

Chemical Composition of Rail Steel.

The chemical composition stated in the specifications for the respective rail sections is calculated :

1. To produce sound ingots under good manufacture ;
2. To provide definite physical properties in the rails to meet the conditions of service of the different sections, as engineering structures.

Ingots for Rails.

Ingots of rail steel which contain from 0.55 to 0.75 of 1 per cent. of carbon are of an entirely different character from those of the low carbons mentioned. This becomes emphasized when the metal is sufficiently deoxidized to form relatively pure steel, which then sets dead in the molds, while the volume of hot molten metal as teemed will be 4.5 to 6 per cent. greater than the volume of the steel in the cold ingot.

When the bath of metal is sufficiently deoxidized to form comparatively pure steel, an interior shrinkage cavity starts to form in the ingot, incident to the cooling and setting on the bottom from the stools and sides of the molds, and then caps over on top, inclosing a larger volume of hot metal than would be the case when cold.

This important fact should be remembered in discussing rail steel, for the greater the degree of its deoxidation, the larger will be the difference between the inclosed volume of hot fluid metal in the mold and the cooler resulting set metal, and the still proportionately lesser volume, should the ingot be allowed to become cold before equalizing the heat and rolling. We must deal according to the temperatures with three conditions or stages of the steel :

1. The greater volume of hot molten metal ;
2. The lesser volume of hot set metal ;
3. The least volume of cold metal in the dimensions of the rail sections or other finished products.

The mill practice to control the conditions in Nos. 1 and 2, must be carried out in an orderly and systematic manner as the proper temperatures occur in the cooling and setting of the steel of the ingots.

The heat of the metal is abstracted by the stools and sides of the molds, and the ingot walls set ; then the top caps over

quickly, before shrinkage in the length of the ingot occurs, and, as the hot volume becomes reduced by the setting exterior walls, the interior shrinkage develops a cavity under its top cap when the ingot is set aside to cool.

The contours of the shrinkage cavities in well-deoxidized steel are parabolic in form, and the cavities are of proportionately greater volume and depth in the long type of ingots than in those which are short and stubby, but of larger volume and base.

Major and Minor Portions of the Shrinkage of Ingots for Rails.

The study of the lessened volume of the cold rail steel ingots from the teemed molten metal shows that the shrinkage consists, favorably for our purpose, of major and minor portions, and that to produce pipeless ingots for rails, we are concerned only with the minor.

The major portion is the exterior shrinkage of the dimensions of $\frac{3}{16}$ to $\frac{1}{4}$ in. per foot of the ingot from the molten state, to hot set metal, and then to cold steel. The roll designer makes provision for this by nearly $\frac{1}{50}$ larger hot templets for the passes in the rolls, to provide for the shrinkage of the hot set metal to the cold dimensions of the finished products.

The minor portion is the less proportional interior shrinkage of the walls of the setting ingot, due to their retaining a temperature just below freezing, while the center of the ingot remains molten for intervals varying from a few minutes in small sizes, to an hour or more in the larger ingots used for rails.

The interior shrinkage, or minor portion, varies with the size and weight of the ingot, but is under 1 per cent. of the volume from hot molten to cold metal in the short, stubby ingots of the weight, size, and type shown in Fig. 1.

These ingots range from 28,600 to 29,000 cu. in., and when set aside to cool, the exterior shrinkage, or major portion, seems to range from 1,150 to 1,200 cu. in., compared to the interior, or minor portion, of only 250 to 300 cu. in. in the cut ingot measured.

The large rail ingots, as soon as capped on top, stripped, weighed, and charged into the reheating furnaces to equalize their own initial heat, must then be bloomed to prevent and limit the major and minor shrinkages to small percentages

before the blooms are finally rolled into finished products. This is illustrated by the cavity of 250 cu. in. in Fig. 1, of the cold cut ingot, compared to the small cavity of only 17 cu. in. in the discard of the bloom, Fig. 2, from a companion ingot of the same melt, which was not set aside to cool, but bloomed from its own initial equalized heat.

Pipeless Ingots for Rails.

Three methods of procedure are possible to prevent the formation of a shrinkage cavity or provide for its subsequent elimination:

1. The customary mill practice required is to strip the ingots and charge them promptly into the reheating furnaces, and then bloom them under their own initial equalized heat;
2. To compress the walls of the ingots, in order to close the slight interior shrinkage cavity which may start to form while their centers are still molten, then reheat and bloom the ingots;
3. To feed the lessening volume of the ingots with molten metal, then reheat and bloom them.

The 70,000,000 tons of rails used in this country were mostly made by the first method.

The first Bessemer ingots were teemed and set aside to cool for inspection and examination, as in crucible steel practice. Many piped rails were subsequently found in the track, and the mill custom of cooling the ingots was abandoned for that now in general use, of stripping the ingots, weighing, and then charging into the reheating furnaces and blooming with their own initial equalized heat.

The ingots of the various sizes were smaller on top than at the base, for stripping, and were top poured, while the walls of the molds were made the thickest on the bottom, to hasten the cooling of the ingots from the base and sides upward to the top. The second and third methods have been used for ingots of crucible and tool steel, and to date, 1912, are adapted to the commercial sizes for rails, although they have not been extensively used for that purpose.

The inverted form of ingot, of which the top is the larger, contributes to the easier escape of any deoxidation products in the molds, and also has a tendency, by cooling more rapidly from the bottom, to keep the shrinkage cavity near the top,

which is required when the third method of procedure is used for teeming. The inverted form of ingot has several advantages over the present form, and may be used in the future for rails, should it make sounder ingots and prove of economical advantage.

Deoxidizers.

The exterior blow-holes in the outside walls of the ingots can be prevented from forming by sufficient deoxidizers, as silicon, ferro-titanium, or their combination, and aluminum. The latter has been extensively used, but all of its oxidation products do not always escape from the metal, and it should not be used when the steel is to be subjected to the present heavy wheel-loads.

The silicon content for rail steel now ranges from 0.10 to 0.20 per cent., to make it sound and prevent small blow-holes from forming in the setting metal.

When sufficient deoxidizers are used to purify efficiently the steel, then, as must be expected, a small cavity starts to form in the top under the cap of the ingot in the setting steel, as already described, and its development should be retarded by stripping the ingot and promptly charging into the reheating furnace.

Rail ingots are no longer held until they become cold before being charged into the reheating furnaces for blooming.

Size, Length, and Weight of Ingots.

The size and length of the ingots must be taken into consideration, for in those ingots of which the length is from four to five times the width of the base, the steel will set on the interior walls long before their vertical shrinkage of hot to cold metal has occurred, and this increased length will add proportionately to the volume of the interior piping or shrinkage cavity.

Split Heads and Piped Rails.

It was customary a few years ago to teem ingots which were only 18 by 20 or 19 in. square, and roll four lengths of 100-lb. 33-ft. rails. The height compared to the base was so great, that before any shrinkage occurred in the vertical hot ingot walls, the interior shrinkage cavities grew so large, they could not be entirely prevented from forming, even by prompt charging of the ingots, after stripping, into the reheating furnaces to equalize the heat for rolling.

The 33-ft. 100-lb. rails rolled from four-rail-length ingots of the long type developed in the track a great many split heads and some true pipes, the product from two or three mills being quite pronounced in this respect. Rails which were rolled during August and September, one purchaser removed in less than six years' service more than 10 per cent. of the order, for split heads. The trackmen would report these rails as piped, for the segregated metal in the head would crack under the fillet and admit the air, which would soon discolor the interior surface, and these were considered as piped rails. There was in some instances a true pipe or shrinkage cavity when rolled, which extended into the center of the web, and well up into the head. The trackmen, however, were not able to distinguish between the true piped rails and the split heads, and it was some time before the latter were attributed to segregation and slag inclosures, which, when recognized, were nearly prevented in subsequent manufacture of rails.

When the mills began to make 33-ft. rails and teemed them in the same ingot-molds which had been used for 30-ft. rails, and then rolled them in four 33-ft. lengths for 100-lb. rails, a great many ingots were not stripped, weighed and charged into the reheating furnaces with sufficient promptness to prevent a number of piped rails, as the requisite mill practice to check them was not then comprehended under the changed conditions of manufacture.

The segregation was also large, and in 1908, for the New York Central Lines, I confined the rolling of Bessemer and open-hearth rails in the United States mills to three 33-ft. rail-length ingots for those of about 19 in. square upon the base. It was also stated in the specifications for the New York Central Lines that short, stubby ingots of from 2.5 to 3 times the length of the width of the base were required for rails. Ingots of about 8,200 lb. weight, teemed in molds 20 by 24 in., have been extensively made for six lengths of 33-ft. 100-ft. rails, and, in good mill practice, with practically complete elimination of piped rails. The blooms, however, are cut, and only rolled in three rail-lengths at a time. Ingots, 25 by 30 in., of about 12,000 lb. weight, have been used for eight 33-ft. basic open-hearth 100-lb. rails where the ordinary rail-mill equipment had not been installed. The ingots were bloomed and then shipped

to a rail-mill to be reheated and rolled, and but a few piped rails were found during manufacture. The rails in the track fulfill the requirements of safety and severe service.

The large mass of metal in the short ingots does not quickly cool, and from the teeming of from 60- to 80-ton melts, the ingots would be charged into the reheating furnaces in 1 hr. and 30 min., and before all the interior metal had set, with but a trace of a shrinkage cavity started.

The distance traversed by the ingots on their cars from the open-hearth department to the strippers and then to the reheating furnaces, aids to consolidate the hot metal in the center of the moving ingots.

Cutting Cold Ingots to See the Volume of the Shrinkage Cavity.

Ingots have been teemed and stripped in the ordinary manner, then taken to the reheating furnaces, and, when ready for rolling, taken out, set aside to cool, and when cut, have shown a shrinkage cavity, as would be expected. The blooms from the companion ingots, when promptly charged into the reheating furnaces and rolled as in proper mill practice, would show only a small trace of the cavity compared to that in the cold cut ingot.

I do not know where similar comparisons will be found outside of my own work of cutting ingots which have been set aside to cool, and also cutting blooms of companion ingots as rolled under the best practice of to-day.¹

Mill Practice.

The necessary mill practice to secure pipeless rails must be understood, and the time limit from teeming, stripping, weighing and charging into the reheating furnaces, involves definite relations to the mass of metal, its chemical composition, the length of the ingots and size.

I call attention in the specifications of the New York Central Lines as to mill practice, that, as soon as the ingots are stripped, they shall be charged into the reheating furnaces to prevent the setting steel cooling from its molten temperature to that of cold metal, and thus avoid the formation of the full shrinkage cavities in the ingots. It has been shown by cutting

¹ *Trans.*, xxxix., 818 to 850 (1908); xl., 821 to 831 (1909).

open a large number of blooms that, by this method of good mill practice, it has been possible to prevent the formation of a shrinkage cavity in the top of the ingot of not more than $\frac{1}{20}$ to $\frac{1}{30}$ of the size which would have occurred, had the ingot been permitted to become completely cold before it was charged into the reheating furnace for rolling.

Bloom crops which have been split from companion ingots show only a slight indication of a shrinkage cavity, which is removed by the discard. The mill practice of promptly charging ingots, after stripping and weighing, into the reheating furnaces, I have followed for many years, and but a few piped rails required rejection during the manufacture, testing and inspection. The latter commenced in the converting department and ran through the mill to the finishing and shipping departments.

There are only 25 piped rails known to have been found subsequently in service in the track in 65-, 70-, 75-, 80-, 95-, and 100-lb. sections out of about 1,100,000 30-ft. rails, of which the length of the ingot was not over, but under, three times the width of the base. The ingots were all stripped by hand in the teeming pit and charged into horizontal reheating furnaces, a mill practice long since abandoned. Many of those rails, containing 0.06 per cent. of phosphorus and from 0.60 to 0.65 per cent. of carbon, are still in freight and branch-line service. Some split heads have developed in these rails, due to segregation and the heavy service to which they have been subjected.

Position of the Rails in the Ingot.

I was at the mills co-operating in the manufacture and inspection of the rails, and commenced in 1893 to indicate their position in the ingots, and stamped on the web of the top, middle, and lower rails the letters *A*, *B*, *C*, respectively. This was for the purpose of studying their subsequent wear and action in the track, which has been so instructive that the practice of designating the rails by letter for identification in the track has become general for the United States.

The *A* rails contained a larger percentage of oxides, which rose in the ingots before complete setting of the steel, and wore faster than the *B* or *C* rails under the same traffic. The break-ages, however, have been slight in either the *A*, *B* or *C* rails after their many years of service.

The ingots were teemed with sharp corners in the molds, of about 2.5 in. radius, and in the *A* rails particularly, oxides and slag were entrained in the corners by the columnar structure of the setting steel.

The gauge-side corner of the *A* rails would show indications of breaking down and spawling to a greater extent under the heavy traffic than the *B* or *C* rails. It was possible, after the rails were in the track 8 or 10 years, to identify by casual inspection the *A* rails from the *B* or *C* rails, by the more frequent spawling on the gauge-side corner of the head.

The *A* rails in the 80-lb. sections which were in the freight tracks of the New York Central & Hudson River Railroad in places of heavy traffic, lost the metal faster than the *B* or *C* rails, and in a few places, were removed and laid by themselves to insure equal heights of the worn surfaces. The same characteristic of wear was noticed on the 100-lb. rails, and also upon the 95-lb. rails of the Boston & Albany Railroad, though the *A* rails were never separated on account of the increased wear.

The comparatively few piped rails indicated that attention to the mill practice of charging the ingots promptly into the reheating furnaces, as already described, did contribute to the soundness of the rails. Therefore, similar good mill practice would be beneficial for the larger and heavier ingots of recent manufacture, and the exceptional freedom from breakages of rails made from them, in service the past winter, has again confirmed the value of such mill practice and essential study for basic open-hearth steel.

Co-operation of Consumer and Manufacturer.

We must first provide the hot molten metal with a chemical composition which will produce sound steel and definite physical properties in the finished product. Our first effort is to secure a well-deoxidized steel, and by proper mill practice make sound ingots. This method has given excellent results in the past, and by the necessary study and attention of the consumers and manufacturers, it is possible to obtain greater excellence. It is important to understand and observe the principles which contribute to better practice, and in this the consumer must materially aid the manufacturer by advising him of the types of failures which occur in rails under service.

It is requisite for more than one person to know the principles of good mill practice, that all of the general conditions of manufacture may be followed to produce the desired results.

There are several methods in which a refractory brick is placed upon the top of the ingot mold, then, when filled with molten steel, is kept hot by gas-jets or charcoal to cause the hot fluid metal to fill up the shrinkage cavity formed by the setting steel in the ingot. Such methods have not been used for rail steel to any extent in America, though many thousand tons have been made from ingots in which coke dust was thrown on the top of the molten steel in the mold, and it fed the shrinking volume of setting steel. Granulated slag has also been used.

When refractory bricks are used, more deoxidizers are added to the steel for the purpose of insuring the elimination of all blow-holes and to increase the tendency of the steel to shrink and permit the molten metal to feed the lessening volume.

It was a common practice a few years ago, in teeming ingots, to add 2 or 3 oz. of aluminum per ton in the molds to reduce the oxides in the steel.

Benjamin Talbot, of England, at the present time, proposes to cast the molten metal in large ingots, 25 in. square or 20 by 24 in. on the base, and poured to a depth of 60 in. or more, and after the ingots are teemed, stripped, and put into the reheating furnaces for about 35 min., and while the center is still molten, to cog them down to 18 in. square. Then the bloom with the fluid center is put into the reheating furnace to set and the heat equalized before it is rolled into the final section. The shrinkage cavities are said to be completely closed, and ingots which have been made in England show on the exterior a chemical composition of about what the ordinary ladle-tests indicate. Then there is a layer of higher carbon content, with a softer center than the outside. He uses a large percentage of aluminum or silicon or ferro-titanium to make his steel sound, and then by his method of precogging, eliminates the cavity and produces an ingot in which the metalloids are not uniformly distributed. Several ingots have been made, a few rolled into rails and tested under the drop in England, which were said to show good results.

The distribution of the soft and hard layers of steel in the rails, as above described, is almost a reversal of the usual segre-

gated rails by the hard center in the head and web being replaced by softer metal than the average for the section.

These layers of hard and soft metal may not be separated under one or more blows of the tup, but are liable to be, under the thousands of blows of the passing heavy wheel-loads of the long and fast trains in climatic temperatures of extreme heat and cold.

Each wheel-load adds but a small increment to the spreading metal in the bearing surface of the head of the rail or the wheel-tread, but eventually it produces a detailed failure in non-homogeneous metal in either case.

The present effort in the United States is to secure as homogeneous metal and fabrication, for railroad service, as possible.

Sir Robert Hadfield recommends his patented process, by which he places a refractory inclosure on the top of the mold and fills it with molten metal, thus making a sink head, filling up the ingot. Then, on top of the molten metal, he places a layer of slag, 0.5 in. or more in thickness, and, on top of that, he puts charcoal and fires it by a blast of air, which prevents setting of the top metal, and feeds the shrinking volume of that beneath. He states his process enables him to produce ingots of which 92 per cent. of the weight is available for use.

It is important to call attention to the fact that both Mr. Talbot and Sir Robert Hadfield consider it necessary to use a larger percentage of deoxidizers than customary in the bath, to produce what they term "Piping Steel." This indicates that their ordinary open-hearth product as made is not of itself sufficient to reduce the oxides in the steel as low as they consider essential to eliminate the exterior blow-holes and make the ingots set solid, except what would be the natural shrinkage cavity which their special methods are intended to prevent or close by treatment.

The views of Mr. Talbot and Sir Robert Hadfield are old, as to the desirability of completely eliminating the blow-holes and causing the steel to set sound at the risk of producing a shrinkage cavity, which must be checked from full development, and have been held and practiced by me for the past 30 years in the production of ingots for steel rails. The deoxidizers, aside from the manganese, should be sufficient to cause the steel to set sound, as is shown in the metal of the cut ingot, Fig. 1, without blow-holes nearly to the extreme top of the cold ingot.

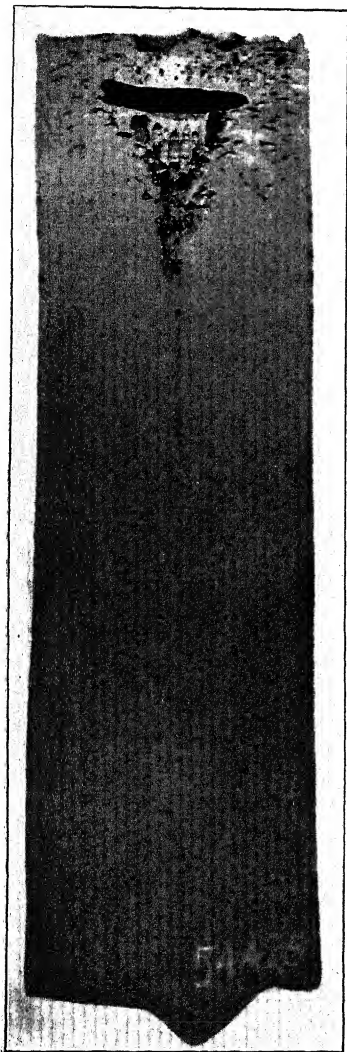


FIG. 1.—CUT AND SPLIT COLD INGOT FROM MELT 54,428.

The major shrinkage is approximately 1,150 cu. in., and the minor 250 cu. in.

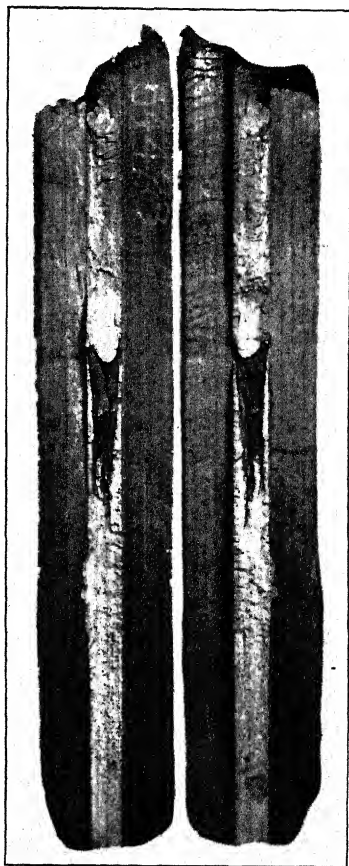


FIG. 2.—BLOOMED CROP FROM COMPANION INGOT OF SAME MELT WHICH WAS CHARGED AND BLOOMED UNDER ITS OWN INITIAL EQUALIZED HEAT.

The shrinkage cavity was 17 cu. in., which is larger than the average from the same class of ingots.

The cavity, as will be seen, was removed in the ordinary discard from the bloom.

Principles to Make Proper Steel and then Sound Ingots.

The principles of making good steel for all high-carbon ingots are understood by metallurgists and manufacturers as well in America as abroad. It is now recognized that the bath must be efficiently purified to secure sound steel of the requisite physical properties and ductility in the fabricated articles or rolled rails. The important fact, however, is not understood by all railway engineers, that the effort to make piping steel is for the purpose of securing sound ingots. The suggestions of Mr. Talbot and Sir Robert Hadfield, to use a large percentage of aluminum in the ingots to reduce more completely the oxides, I do not consider advisable, from the difficulties already experienced with aluminum so used in rail steel for our heavy wheel-loads. It would be better to use silicon or a combination of silicon and ferro-titanium to secure the desired results. We do not use as high percentages of silicon in steel as are employed abroad, except for tires.

It is now found for our heavy wheel-loads and severe service in the low temperatures of several of the important trunk-lines, that the high-silicon tires break more frequently than those in which the content is lower. The suggestion to use from 0.3 to 0.4 per cent. of silicon in rail steel without modification of the other chemical constituents, would involve the risk of many rails breaking from the slipping of the drivers upon the rail-heads. We must proceed with proper caution in introducing deoxidizers which remain, or their oxidation products are liable so to do, in the bath of steel. Ferro-titanium, while more expensive than either aluminum or silicon, also acts as a flux, and can be used without danger of leaving its oxidation products in the well-made bath of steel.

The impression prevails that to produce a greater soundness in steel by the use of ferro-titanium as a deoxidizer is detrimental rather than beneficial, from the increased tendency to pipe the ingots. It is the proper use of subsidiary deoxidizers and their attending conditions which must be understood to secure sounder steel with, rather than without them.

Segregation of Basic Open-Hearth Steel Ingots.

The segregation of Bessemer ingots has been studied extensively, but the basic open-hearth ingots for steel rails and wheels have not received requisite attention. I have studied their seg-

regation in several ingots, but do not find it as great in well-purified steel as might be expected from Bessemer, which contains two or more times the impurities of phosphorus and sulphur. Well-melted, purified basic open-hearth steel sets quietly and the segregation becomes less in degree.

The Illinois Steel Co., at Gary, when rolling rails for the New York Central Lines in 1912, at my request took one ingot weighing 8,100 lb. from melt No. 54,428 and charged it into the reheating furnace as in ordinary mill practice. Then, in about 2.5 hr., when in condition to roll, drew and set it outside the furnace to cool. The ingot was 20 by 24 in. on the base and poured 71 in. long.

The shrinkage cavity shown in the cold cut ingot, Fig. 1, is fully developed from hot to cold steel, and is more than 15 times larger than in the bloom-crop of the rolled companion ingot as charged in the usual mill practice. Charging the ingot 10 or 15 min. earlier would have prevented even as large a shrinkage cavity as found, in the bloom-crop, though it shows that 1.4 per cent. of sound metal was cut off in the usual discard and 0.75 in. in depth was planed from the center of the bloom and the small cavity was entirely removed.

The crop from the 8-by 8-in. bloom, 46 in. long, from the companion ingot was split to examine the interior shrinkage cavities and segregation; see Fig. 2. The ingot when cool was split and planed, and five vertical rows of holes were drilled in one-half of the ingot, which were marked respectively, *A*, *B*, *C*, *D*, and *E*. The distance between the vertical rows *A* and *B* was 2.25 in., and for the other rows, 3 in. The transverse rows Nos. 1 to 9, inclusive, were 3.75 in. apart, but between Nos. 10 to 15, inclusive, the spacing was 7.5 in.

Drillings were taken from 64 holes of the ingot, and chemical analyses made for the carbon, manganese, phosphorus, sulphur, and silicon, the results being shown respectively in Figs. 3, 4, 5, 6 and 7, the carbon being determined by combustion.

The crop of the companion bloom weighed 759 lb., representing 9.4 per cent. of the ingot, and was split and drillings taken from three vertical rows of holes marked *A*, *B* and *C*. This made 73 holes from which complete analyses were made.

Fig. 3 is the carbon diagram from the ingot and companion bloom.

The chemical composition by the ladle test was: C, 0.71; Mn, 0.83; P, 0.025; S, 0.038; Si, 0.18. The vertical row of holes under *A*, near the exterior of the ingot, shows that the carbon ranged from 0.67 to 0.69. The carbon for the 15 holes of the vertical row *B* shows a lesser content, while the row *C* has a higher content than the ladle analysis from the third to the tenth transverse row. The vertical row *D* is one point above the ladle composition from the fourth to the

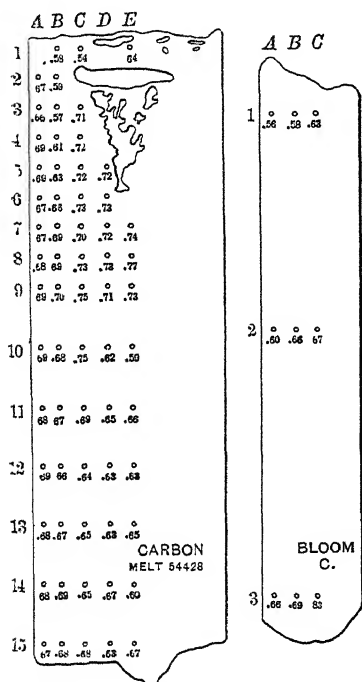


FIG. 3.—CARBON DIAGRAM OF INGOT AND BLOOM.

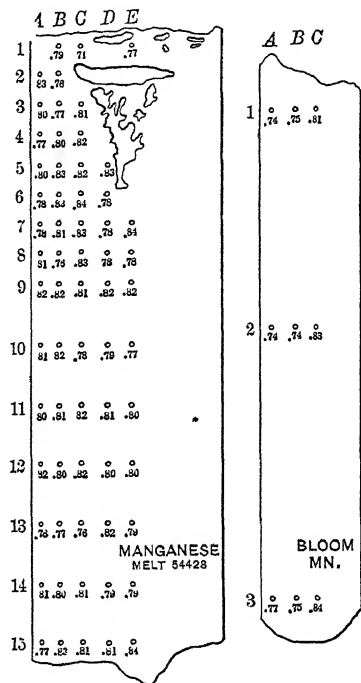


FIG. 4.—MANGANESE DIAGRAM OF INGOT AND BLOOM.

eighth transverse row, while the vertical row *E* shows above the ladle analysis from the seventh to the ninth transverse row. The carbon in the vertical row *C* appears below the normal under the eleventh transverse row, and in *D* and *E*, below the ninth row.

The segregation of the carbon is not as large as would be expected from the size of the ingot, the top indicating that the steel set quiet under the silicon content of 0.18 in the ladle, and that possibly one or two points were subsequently absorbed by the further deoxidization of the steel in the mold before setting.

The carbon in the crop of the companion bloom, Fig. 2, shows in the vertical row, *A*, in the top transverse row, 0.56; in the middle or second row, 0.60, and in the bottom or third row, 0.66, the latter being five points below the ladle analysis. The row *B* is also slightly less, and that of *C*, with the exception of 0.83 in the third transverse row, which is the highest indication of segregation of carbon found. The slightly higher interior grouping of the carbon above the center and a local reduction in the lower part of the ingot, are usually considered

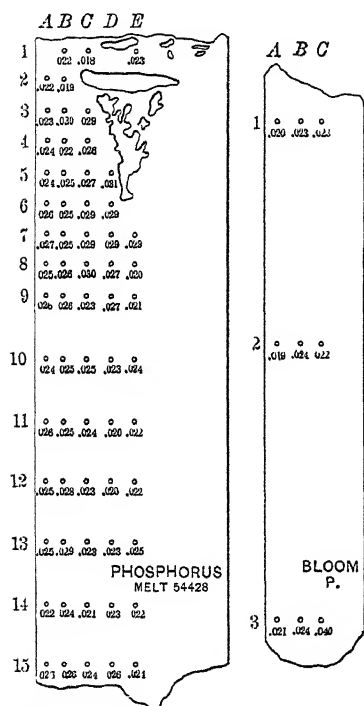


FIG. 5.—PHOSPHORUS DIAGRAM OF INGOT AND BLOOM.

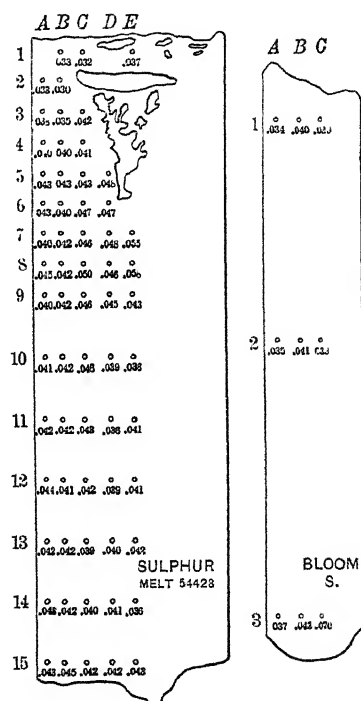


FIG. 6.—SULPHUR DIAGRAM OF INGOT AND BLOOM.

sufficient to explain the variations, but they seem to me inadequate. The average carbon from the drillings is apparently 0.67, while before cooling from the ladle test it seemed to be 0.71, a small practical variation, though it should receive consideration in an investigation of the causes of the segregation.

Metallic titanium to the amount of 0.1 per cent. in the ladle would have reduced the segregation of the carbon to some extent, though with a tendency to increase the pipe, and to obviate the latter it would have been necessary to handle the

ingot more promptly from the teeming, stripping, and charging into the reheating furnace.

Fig. 4 is the manganese diagram, which has a nearly uniform content. Fig. 5 is the phosphorus diagram, in which the segregation is unusually small, and only reaches 0.30 in one drilling, and 0.40 in the bloom. (See line of holes in the third transverse row.) Fig. 6 shows the sulphur diagram, in which there is some segregation; nearly all contents are slightly higher than the ladle analysis. Fig. 7 is the sili-

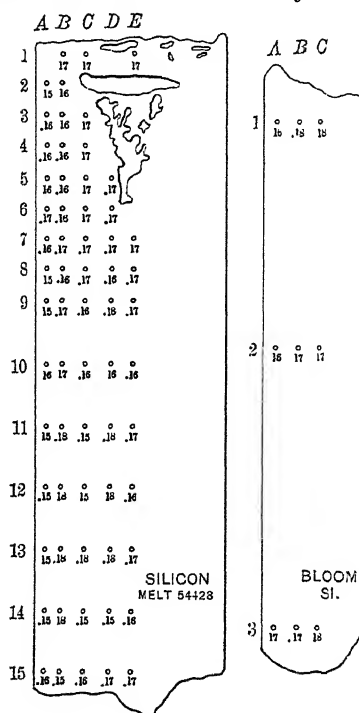


FIG. 7.—SILICON DIAGRAM OF INGOT AND BLOOM.

con diagram for the ingot and bloom, and is nearly uniform, and while it is one or two points under the ladle analysis as a rule, it is not above in a single instance in the diagram of the ingot or bloom. A number of similar diagrams will enable us to study more specifically the causes of segregation.

A 12,000-lb. basic open-hearth ingot with a greater percentage of deoxidizers shows less apparent variation in the carbon, but more in the phosphorus, though starting with a lower ladle content.

It is of equal interest to report the segregation found in two engine truck axles from an acid open-hearth melt of 1911 make, which failed under a light but short service. The ladle analysis reported by the makers was: C, 0.443; Mn, 0.52; P, 0.034; S, 0.022; Si, 0.028 per cent.

The analyses of the outside and center of the 6.25 in. in diameter finished axle were:

Outside: C, 0.458; Mn, 0.495; P, 0.031; S, 0.018; Si, 0.014.

Center: C, 0.801; Mn, 0.591; P, 0.103; S, 0.064; Si, 0.018.

The segregation of all the elements is excessive except silicon, which was too low to quiet the steel and make it set dead in the mold. The ingots were bottom cast.

The Drop and Exhausted-Ductility Tests for Basic Open-Hearth Steel Rails.

The exhausted-ductility tests for the purification of the steel were introduced into the specifications for the New York Central Lines in 1910 to secure from the preceding 18 years' experience with the elongation tests of Bessemer rails, those of sufficient toughness to withstand the low temperatures in the Mohawk valley of from 20° to 30° below zero, though 40° is often reported by the trackmen.

The ductility which is possible for a given composition, size of ingots, section, and other steps of manufacture, has been practically ascertained, and it is to see by the exhausted-ductility tests whether or not it is secured.

It requires but a moment to stamp the crop of the rail with the 6-in. spacing bar of seven points before placing the butt on the supports of the drop-testing machine. The elongation of each of the six marked inches on the test-butts after the drop is measured by a flexible rule, and the increase in hundredths of inches per inch indicates the per cent. of elongation.

It takes but a moment to test the butt, and the exhausted ductility is obtained in three or more blows, though it is measured after each blow, and recorded. The lower carbon content of the specification gives one range and the maximum content a higher range, and the mean carbon content is between the two. Either one of the three can be used as may be necessary in locations for safety, speeds, and wheel-loads of service.

The Sections 1, Chemical Composition, and 4, Drop and Ductility Tests, are here incorporated as part of the paper.

NEW YORK CENTRAL LINES.

Specifications for Basic Open-Hearth Rails.

1st—CHEMICAL COMPOSITION :		80-lb.	90-lb.	100 and 105 lb.
Carbon.....	from	0.55 to 0.68	0.60 to 0.73	0.62 to 0.75
Manganese .	from	0.70 to 1.00	0.70 to 1.00	0.70 to 1.00
Silicon	from	0.10 to 0.20	0.10 to 0.20	0.10 to 0.20
Phos. not exceed.....		0.04	0.04	0.04

To adjust the chemical composition to the special conditions of manufacture at each mill, the engineer representing the railroad company, from the inspection of the ingots, the heating, blooming, and rolling into rails, shall have the right to select the lower or average limit of either the silicon or manganese, or both, with the average carbon content as the working basis for making the steel, as he may find requisite for good setting ingots with freedom from pipes and rolling into tough steel by the plant of the manufacturer.

4. Drop and Ductility Tests.

A drop-test to be made of a crop from the top bar of the second, the middle, and the last full ingot of the melt. The crop 4 to 6 ft. long to be stamped with a spacing bar of six inch spaces on the base, head or side as desired.

Each butt of the three tests must show under a blow of the drop of 18 ft. for the 80-lb. or 90-lb. section, and 20-ft. for the 100-lb. section, at least 6 per cent elongation for 1 in. or 5 per cent each for two consecutive inches before fracture, for acceptance of the melt.

Should any test-piece under the first blow of the drop not break, but fail to show the percentage of elongation specified, the test-piece shall be subjected in the same position to a second blow and the results so obtained govern in passing the test.

The ductility of at least one specimen of each melt to be exhausted by one or more blows of the drop, and a record made of the respective elongations of each test.

The crop or butt is liable to be chilled accidentally in entering the rolls several times, or it may be caused by other delays, and should it break under a single blow without showing the percentage of elongation specified, it shall be considered as indicating deficient ductility or chilled metal, and the results must be rejected.

The Engineer-of-inspection representing the Railroad Company must then take a duplicate test from the same ingot at the top end of the *A* or *B* rail, according to the nine or greater percentage of discard, and the results taken in place of those from the first crop or test to determine whether or not the piece had the requisite ductility in accordance with the specifications.

The distinction between a chilled test-crop and those of inadequate ductility must be ascertained according to the above prescribed tests before rejections are made or rails accepted.

The drop testing machine shall have a tup of 2,000 lb. weight, the striking face of which shall have a radius of not more than 5 in. on solid supports, centers 3 ft. apart, for the test butts. The anvil block shall weigh at least 20,000 lb. and the supports shall be part of or firmly secured to the anvil. The report of drop test shall state the atmospheric temperature at the time the test was made. The testing shall proceed concurrently with the operation of the mill. The temperature of the test butts, to be between 40 and 100 degrees.

NOTE:—The drop for the 105-lb. rails should be 21 ft., as a second blow is often required to give the elongation.

The specifications provide for the distinction between chilled butts and those of inadequate ductility, one of the first essentials in the tests of high-quality rails.

The term Ductility in the specifications for the New York Central Lines, is used in the sense of tenacity and toughness of the steel, the exhausted ductility being its measure. This also becomes a soundness test for seams, segregation, slag inclusions and other foreign matter in the web or head of the rails, and is better than the special nick test in other specifications.

The Elongation and Exhausted-Ductility tests of steel rails are made concurrently with their manufacture at the plant under the drop-testing machine on 4- or 5-ft. lengths of the top crop of the rail bars, the results being tabulated as shown by the appended blanks for ductility and elongation tests, a copy being furnished to the manufacturers.

The butts are tested within 3 hr. or less after the ingots of a melt are teemed and rolled, and the facts as to the full ductility of the steel as made and rolled are available for the manufacture of subsequent melts. The utilization of such knowledge of each melt has enabled the manufacturers to make rails of greater uniformity, homogeneity, safety, and economy of the output than was believed possible before such tests were installed, as illustrated by the results shown in the appended sheets which give the dates of manufacture. The practical non-breakage of such rails in the track for the winter of 1911-12, shows a high quality was secured in their manufacture.

When the ductility tests were first proposed, the most experienced makers did not think it possible to secure such uniform results per melt of steel. It was considered by some of the presidents of the steel mills that, dealing with the larger melts of from 60 to 80 tons in a ladle, the temperature might be just right for teeming the first ingot, but too low for the middle ingot to give the full ductility of the chemical composition and to reduce its percentage for the last ingot teemed of the melt.

Uniformity of Output.

The heavy body of slag over the top of the molten steel in

the ladle did not allow sufficient heat to escape to modify the ductility in well-made steel from the first to the last ingot poured.

The manufacturers were without any previous knowledge of the full ductility which could be obtained from the different melts and did not expect that it would much exceed that stated as the minimum requirements for acceptance, and they were surprised and gratified with the results. The exhausted-ductility tests are made in rotation from ingots of the different melts, exhausting one from the second ingot poured in one melt, then from the middle ingot in the next melt, then from the last ingot poured in the next melt, when rotation again commences of the ingots to be tested in succeeding melts.

The examination of the results from more than 200,000 tons of steel now proves that it is possible to secure nearly a uniform output from each melt. It is found that there is some difference in the ductility which may be obtained, according to the section which is being rolled. The well-designed sections of not too great width of base compared to the height, finish with nearly the full ductility of the metal.

Five sheets of ductility and elongation tests of two melts each are given, three for the Lackawanna Steel Co. for 80- and 100-lb. Dudley sections, including the 105-lb. reinforced web and base by fillets of 1-in. radius, and two sheets for the Illinois Steel Mill at Gary, for 90- and 100-lb., A. S. C. E. sections. The detailed information on them about the drop and size of ingots, melt, and number of rails, is sufficiently explicit without further explanation.

The average chemical composition for 80-lb. rails is intended to produce 21 per cent. of ductility, for the reason that in many places they carry the same class of engines and are subjected to heavy traffic in temperatures of 30° to 40° below zero. It required a short experience for the melters of the different furnaces to become familiar with making the steel of such a degree of purification as practically to obtain the ductility due to the chemical composition.

20,500 tons of 80-lb. rails were rolled in the Dudley 5½-in. section in 1911, of which the computed ductility for the average range of carbon was 21 per cent. per inch. The average ductility for the tonnage shipped was 19.8 per cent.

Application of Knowledge.

This illustrates the value of the complete exhausted-ductility tests per melt, for it is possible in each to trace the specific conditions of the furnace, the melting scrap used, etc., and bring each melt up to nearly the same degree of purification of the bath. This deals directly with the making of the steel and furnishes information to the manufacturers never before accessible, and enables them to produce a more uniform product from the different melts. This is evident by the sheets of the ductility tests, for melt No. 7,802 gives 21 per cent., nearly all which could be obtained for the carbon and manganese, while for melt No. 12,279, 18 per cent. is good for the carbon and manganese, though 20 per cent. is possible. Melt No. 12,900 in the 100-lb. is 26 per cent., which is above the average, for the manganese is low and the base necked before fracture, while for melt No. 6,649, 22 per cent. is good for the carbon and manganese. Melt No. 10,002 has 18 per cent. ductility for carbon 0.704, and manganese 0.88, which is good, while melt No. 11,919, for the upper limit of 0.75 carbon and 0.80 manganese, has 13 per cent. ductility, the bars being rolled slightly colder than usual, as the permanent set is only 1.1 in., and the tests of the 2d and 32d ingots did not give the required elongation for one blow of the drop, and a second was required.

The melts Nos. 46,031 and 34,029, rolled by the Illinois Steel Co., Gary, gave respectively 13 and 12 per cent. in the 90-lb. A. S. C. E. Melts Nos. 43,248 and 45,247 gave respectively 19 and 17 per cent. elongation.

The 90-lb. A. S. C. E. section has not as yet given as uniform ductility or as high a range as most of those sections. The tabulated results of the ductility and elongation tests teach that after the distinction between the chilled crop and deficient ductility is ascertained, the steel runs for most of the mills very uniform per ingot.

The manufacturers have examined the tabulated results of their product and some have adopted the system of testing and keeping the same records for all rails made for their own information, whether requested by their customers or not, as is demanded by the New York Central Lines.

The American Railway Engineering Association has followed the New York Central Lines and adopted nearly their ductility and elongation tests in its specifications.

This is to provide for the great progress in transporting the heavier wheel-loads under the increased speeds of the past two decades. The climatic conditions of 1911 above the 40th degree parallel of north latitude were unusual, as there was a daily excess of temperature of from 2° to 4° above normal at most of the United States Weather Bureau stations on the New York Central Lines, at Chicago reaching $1,262^{\circ}$ of accumulated temperature above normal for the year.

The warm isotherms extended to higher latitudes than usual, followed at Chicago by the coldest January yet known, while the months of February and March were unusually cold. Embankments were frozen from 4 to 7 ft. deep and two to three times the usual number of Bessemer rails were broken, while only one of the basic open-hearth rails made under the ductility and elongation tests of the specifications of the New York Central Lines, broke in from 9,000 to 10,000 rails in the same service, with the exception of the 90-lb. A. S. C. E. section.

Adequate System of Tests During Manufacture.

It is important to remember that when a product of certain physical properties is desired from the manufacturer, some adequate system of tests must be applied to see that the quality of the material is made during the process of manufacture, for any subsequent inspection will not add quality.

The tests of each melt are made and the results known before another melt is tapped from the same furnace, and consequently greater uniformity of the product can be maintained in the general output.

The consumer must aid the manufacturer in prescribing and making the tests to secure the highest possible product from the mill, which is as advantageous to the manufacturer as to the customer. The objections have not arisen that were urged by some persons against the adoption of the ductility tests, stating that oftentimes measurements of 0.01 in. would be the cause of many disputes between the customer and manufacturer. The question of a lessened ductility does not occur in the majority of the melts. The output is so much better than the requirements of the specifications that these disputes did not arise.

The melters and every one connected with the manufacture of the steel take pride in their particular output, and the result

is a decided improvement in the skill and manufacture of every melt of steel.

This of itself tends to increase the factor of safety and is of service to the entire community.

*Conclusions for the Manufacture of the Present Basic
Open-Hearth Rails.*

1. The chemical composition should provide for sound steel of ample physical properties of tenacity and toughness rather than hardness combined with brittleness.

2. The impurities, phosphorus and sulphur, should be of minor content so the bath of metal can be purified to produce the large percentage of toughness and ductility due to the specified chemical composition.

3. The ingot should have such relations of area of base compared to the height and weight that under good mill practice and suitable deoxidizers, it can be made with controlled segregation, and only a trace of a shrinkage cavity in the top; then, when bloomed under its equalized initial heat, it is rendered pipeless by the usual 8 to 10 per cent. discard.

4. Ferro carbon titanium can replace aluminum completely and silicon partly, as a deoxidizer, with advantage, to purify and check the segregation in rail, tire, and axle steels, and also some of the lower grades of carbon steels where great purity is desired.

5. The ductility and elongation tests to date (1913) furnish the best and only prompt means of determining the degree of purification of the steel per melt as it is made by indicating the physical properties secured before another melt is tapped from the same furnace, and are of decided advantage to the manufacturer as well as to the consumer. These tests are so advanced that they must be applied with knowledge and understanding for proper results, and not made mechanically for specified records.

6. Every process or step of the entire manufacture of the steel, rolling and finishing of the rails, must contribute its part to secure the highest quality of the product incident to the chemical composition.

7. Specifications should be drawn to indicate some of the major necessities of the consumer, and the tests and inspection conducted in a spirit to aid and invite the co-operation of the manufacturers to meet the progressive requirements in rail steel.

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Form C E O. No 1

OFFICE OF CONSULTING ENGINEER, RAILS, TIRES AND
STRUCTURAL STEEL.

Grand Central Terminal.

New York.

DUCTILITY AND ELONGATION TESTS.

Sheet No. 1.

Basic Open Hearth Rails 80 lbs. per yd 801 Section. Moment of Inertia 28.5
4th Power inches. Manufactured by Lackawanna Steel Co., for N. Y. C. & H R R.
Date Rolled, 7-12-12 Day. Ht. of Drop 18 ft. Tup 2,000 lbs., Supports 3 ft.
Centers. Ingots, 19 by 19 inches. Teemed in 4 Rail Lengths. Blooms Cut, none,
Reheated, none Rolled in 4 Rail Lengths.

Melt No.	No. of Rails, 120	No. of Ingots, 30	Date, 7-12	C., 655	Mn., 91	Si., 157	Ph., .020	Sul., .033	Cu	
7,802										
Ingot No.	No. of Drop	Permt Set Inches	Elongation Per Inch.						Total	Remarks
			1 In	2 In.	3 In.	4 In	5 In.	6 In.		
2	1	1.8	1.05	1.07	1.08	1.08	1.06	1.05	6.39	No 263
20	1	1.8	1.05	1.07	1.08	1.07	1.06	1.06	6.39	
	2	3.2	1.09	1.14	1.16	1.16	1.13	1.11	6.79	
	3	5.0	1.14	1.17	1.20	1.19	1.16	1.14	7.00	
	4	Sh'rd.	1.15	1.19	1.21	1.20	1.17	1.14	7.06	
30	1	1.8	1.05	1.07	1.07	1.07	1.07	1.05	6.38	

Melt No.	No. of Rails, 148	No. of Ingots, 37	Date, 7-12	C., 645	Mn., 99	Si., 141	Ph., .018	Sul., .046	Cu.	
12,279										
Ingot No.	No. of Drop	Permt Set.	Elongation Per Inch.						Total	Remarks.
			1 In.	2 In	3 In.	4 In.	5 In	6 In.		
2	1	1.8	1.05	1.06	1.08	1.08	1.07	1.05	6.39	No. 264
	2	3.2	1.10	1.12	1.15	1.15	1.13	1.10	6.75	
	3	Broke	1.13	1.15	1.18	1.18	*1.15	1.11	6.90	
20	1	1.7	1.05	1.06	1.07	1.08	1.06	1.05	6.37	
37	1	1.8	1.05	1.07	1.08	1.08	1.06	1.05	6.39	

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OFFICE OF CONSULTING ENGINEER, RAILS, TIRES AND
STRUCTURAL STEEL.

Grand Central Terminal.

New York.

DUCTILITY AND ELONGATION TESTS.

Sheet No. 2.

Basic Open Hearth Rails 100 lbs. per yd. 1001 Section. Moment of Inertia 48.5 4th Power inches. Manufactured by Lackawanna Steel Co. for N. Y. C. & H. R. R. Date Rolled 12-17-11. Ht. of Drop 20 ft. Tap 2000 lbs. Supports 3 ft. Centers. Ingots 19 by 19 inches Teemed in 3 Rail Lengths Blooms Cut, none. Reheated, none. Rolled in 3 Rail Lengths.

Melt No. 12,900	No of Rails, 120	No of Ingots, 40	Date, 12-17	C .715	Mn .74	Si .102	Ph .025	Sul. .040	Cu	
Ingot No	No of Drop.	Permt. Set Inches	Elongation per Inch						Total.	Remarks.
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.		
2	1	1.4	1.04	1.05	1.05	1.06	1.06	1.05	6.31	
	2	2.6	1.07	1.10	1.11	1.13	1.12	1.10	6.63	
	3	3.7	1.12	1.17	1.21	1.20	1.16	1.11	6.97	
	4	Broke	1.12	1.17	*1.24	1.26	1.20	1.14	7.13	P. M.
20	1	1.5	1.05	1.06	1.06	1.07	1.05	1.04	6.33	
40	1	1.5	1.05	1.05	1.06	1.06	1.06	1.05	6.33	

Melt No. 6,649	No of Rails, 117	No of Ingots, 39	Date, 12-17	C .682	Mn .90	Si .115	Ph .018	Sul. .038	Cu.	
Ingot No.	No. of Drop	Permt Set.	Elongation per Inch.						Total.	Remarks.
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.		
2	1	1.4	1.05	1.05	1.06	1.06	1.05	1.04	6.31	
20	1	1.3	1.05	1.06	1.06	1.06	1.04	1.03	6.30	
	2	2.5	1.10	1.12	1.12	1.10	1.07	1.05	6.56	
	3	3.6	1.13	1.17	1.18	1.15	1.12	1.09	6.84	
	4	4.8	1.15	1.20	1.22	1.21	1.15	1.10	7.03	
	5	Broke	1.15	1.20	1.22	1.21	1.15	1.12	7.05	
39	1	1.4	1.04	1.05	1.06	1.06	1.04	1.04	6.30	

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Form C. E. O No 1.

OFFICE OF CONSULTING ENGINEER, RAILS, TIRES AND
STRUCTURAL STEEL.

Grand Central Terminal.

New York.

DUCTILITY AND ELONGATION TESTS.

Sheet No. 3.

Basic Open Hearth Rails 105 lbs. per yd. 1051 Section. Moment of Inertia 49.79
4th Power inches. Manufactured by Lackawanna Steel Co., for N. Y. C. & H. R.R.
Date rolled 1-1-13 Night Turn. Ht of Drop 20 ft. Tup 2,000 lbs. Supports 3 ft.
Centers Ingots 19 by 19 inches. Teemed in 3 Rail Lengths. Blooms Cut, none.
Reheated, none. Rolled in 3 Rail Lengths.

Melt No. 10,002	No. of Rails, 120	No. of Ingots, 40.	Date, 1-1	C .704	Mn .88	Sil. .102	Ph. .020	Sul .033	Cu.	
Ingot No.	No. of Drop.	Permt. Set Inches.	Elongation Per Inch.						Total.	Remarks.
			1 in	2 in.	3 in.	4 in.	5 in.	6 in.		
2	1	1.2	1.03	1.03	1.05	1.05	1.04	1.03	6.23	No. 100 Temp. 48°
	2	2.1	1.04	1.05	1.09	1.09	1.10	1.08	6.45	
	3	3.1	1.06	1.09	1.14	1.15	1.14	1.12	6.70	
	4	Broke	1.08	1.11	1.17	*1.15	*1.18	1.13	6.82	
20	1	1.3	1.04	1.04	1.05	1.04	1.05	1.04	6.26	
40	1	1.3	1.04	1.04	1.05	1.05	1.05	1.03	6.26	

Melt No. 11,919	No. of Rails, 96	No. of Ingots, 32.	Date 1-12	C. .750	Mn. 80	Sil. .125	Ph. .022	Sul. .031	Cu.	
Ingot No.	No. of Drop.	Permt. Set.	Elongation Per Inch.						Total	Remarks.
			1 in	2 in.	3 in.	4 in.	5 in.	6 in.		
2	1	1.1	1.02	1.03	1.04	1.04	1.04	1.03	6.20	No. 101
	2	1.9	1.04	1.05	1.08	1.09	1.08	1.06	6.40	
20	1	1.1	1.03	1.03	1.04	1.05	1.05	1.03	6.23	
	2	1.9	1.04	1.05	1.08	1.08	1.08	1.07	6.40	
	3	Broke	1.04	1.05	1.08	1.12	1.13	*1.11	6.53	
32	1	1.1	1.03	1.04	1.04	1.05	1.04	1.03	6.23	
	2	1.9	1.07	1.08	1.09	1.08	1.05	1.05	6.42	

Note :

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Form C E. O. No. 1.

OFFICE OF CONSULTING ENGINEER, RAILS, TIRES AND
STRUCTURAL STEEL.

Grand Central Terminal.
New York.

DUCTILITY AND ELONGATION TESTS.

Sheet No. 4.

Basic Open Hearth Rails 90 lbs per sq. yd. 9002 Section. Moment of Inertia 34.0 4th Power inches. Manufactured by Illinois Steel Co., Gary, for C. C. C. & St. L. Date Rolled 1-16-13 day turn. Ht. of Drop 18 ft. Tup 2,000 lbs. Supports 3 ft. Centers Ingots 20 by 24 inches. Teamed in 7 Rail Lengths. Blooms Cut, 2 Reheated, none. Rolled in 3 and 4 Rail Length

Melt No 46,031	No of Rails, 154	No of Ingots, 22	Date, 1-16	C. .66	Mn. .79	Sil. .15	Ph. .018	Sul. .057	Cu.	
Ingot No	No. of Drop,	Permt. Set Inches.	Elongation per Inch.						Total	Remarks.
			1 in.	2 in	3 in.	4 in.	5 in	6 in.		
2	1	1.4	1.03	1.05	1.05	1.05	1.05	1.03	6.26	Temp. 46
11	1	1.3	1.04	1.05	1.05	1.04	1.04	1.02	6.24	
	2	2.4	1.06	1.09	1.10	1.09	1.06	1.06	6.46	
	3	Broke	1.08	1.11	1.13	1.12	1.06	1.06	6.56	
22	1	1.4	1.04	1.05	1.05	1.05	1.03	1.03	6.25	

Melt No 34,029	No of Rails, 147	No of Ingots, 21	Date, 1-16	C. .72	Mn. .79	Sil. .16	Ph. .022	Sul. .036	Cu.	
Ingot No.	No of Drop,	Permt. Set.	Elongation per Inch.						Total	Remarks.
			1 in	2 in.	3 in.	4 in.	5 in.	6 in.		
2	1	1.4	1.04	1.05	1.05	1.05	1.04	1.03	6.26	Temp. 46
10	1	1.3	1.04	1.04	1.05	1.05	1.03	1.03	6.24	
21	1	1.4	1.03	1.04	1.05	1.05	1.04	1.03	6.24	
	2	2.4	1.06	1.07	1.09	1.09	1.07	1.06	6.44	
	3	Broke	1.08	1.09	1.12	*1.12	1.09	1.07	6.57	

Note :

P. C. DEMPSEY,
Inspector.

Copying Ink.

Form C. E. O. No. 1.

OFFICE OF CONSULTING ENGINEER, RAILS, TIRES AND
STRUCTURAL STEEL.

Grand Central Terminal,

New York.

DUCTILITY AND ELONGATION TESTS

Sheet No 5.

Basic Open Hearth Rails 100 lbs. per yd. 1001 Section. Moment of Inertia 43.8 4th Power Inches. Manufactured by Ill. Steel Co., Gary, for Mich. Cent. Date rolled June 3, '11. Ht of Drop 20 ft. Top. 2000 lbs., Supports 3 ft. Centers. Ingots 20 by 24 inches Teemed in 6 Rail Lengths. Blooms Cut, 2. Reheated, none. Rolled in 2-3 Rail Lengths.

Melt No. 43,248	No of Rails, 96	No of Ingots, 16	Date 6-3	C 64	Mn 82	Sil .16	Ph. .020	Sul .027	Cu	
Ingot No.	No. of Drop	Permt Set Inches	Elongation Per Inch						Total.	Remarks.
			1 in	2 in	3 in	4 in.	5 in	6 in		
2	1	1.5	1.04	1.05	1.07	1.07	1.06	1.04	6.33	Temp. 70°
	2	2.7	1.07	1.10	1.12	1.12	1.09	1.07	6.57	
	3	3.9	1.11	1.15	1.17	1.17	1.12	1.07	6.79	
	4	Shear	1.13	1.17	1.19	*1.18	1.13	1.08	6.88	
12	1	1.5	1.05	1.06	1.07	1.07	1.05	1.04	6.34	
24	1	1.6	1.02	1.04	1.07	1.07	1.06	1.05	6.31	

Melt No. 45,257	No of Rails, 138	No. of Ingots, 23	Date 6-3	C 69	Mn. .90	Sil. .18	Ph. .026	Sul .033	Cu.	
Ingot No.	No. of Drop	Permt. Set	Elongation Per Inch.						Total	Remarks.
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.		
2	1	1.4	1.03	1.05	1.06	1.07	1.05	1.04	6.30	Temp. 72°
11	1	1.4	1.05	1.05	1.07	1.07	1.05	1.04	6.33	
	2	2.5	1.07	1.10	1.12	1.12	1.09	1.06	6.56	
	3	Broke	1.12	1.17	*1.17	1.16	1.11	1.07	6.80	
23	1	1.5	1.05	1.06	1.07	1.07	1.05	1.05	6.35	

Note :

The Use of Anti-Piping Thermit in Casting Steel Ingots.

BY E. A. BECK, NEW YORK, N. Y.

(New York Meeting, February, 1911.)

For a number of years many attempts have been made to use thermit in order to do away with piping in ingots. Some of these attempts were successful, while others did not give the expected results. Nearly every plant had its own method of using the thermit, and no special care was taken to study the results or to find the causes for lack of success.

Formerly the principle of the process was to decrease the piping by introducing a can filled with thermit, attached to an iron rod, into the head of the ingot directly after it had been poured. In this case the thermit reaction took place in the upper part of the ingot, and it was thought that through the heat produced the head would keep liquid longer and piping thus be reduced. In other words, the aim was to keep the head of the ingot open by making use of the heat of the thermit reaction, which would thereby give the gases in the steel sufficient time to escape.

This result can be attained by the application of a large enough quantity of thermit, but it would not be economical in general practice, and could only be used for high-grade material.

Many steel-makers abroad and in this country have had this experience. The Th. Goldschmidt A. G., Germany, has, therefore, made a special study of this question with the co-operation of a well-known steel company, and has worked out a new method for the application of thermit for this purpose. The results obtained have been most satisfactory and reliable, and this process is now used in Germany on a large scale in many prominent steel-works.

The thermit is applied in cans, as in former years, but the principle of the process is entirely different. The object of using the thermit is not to heat the steel and keep it liquid,

but to create a stirring reaction in the ingot, by which means large segregations, blow-holes, and pipes will be materially reduced, if not practically eliminated. In order to obtain the most beneficial results the can must be introduced as far as possible into the ingot, so that the reaction is practically started at the bottom of the ingot and the stirring action proceeds from the bottom to the top. In a general way this is the operation of the new method through which, by the application of a very small amount of thermit, great beneficial effects are obtained.

The best results were obtained in so-called non-silicon treated material, which is used especially in the manufacture of sheets and plates. Long and carefully conducted tests of this process has been made at the Steel Works Schulz-Knaudt A. G., Angerott, Germany, and are fully described by Dr. C. Canaris.¹ The same author has published² his latest experience with this process. 17,891 ingots, weighing 35,902 tons, have been treated with anti-piping thermit, and the results obtained showed a considerable improvement in the steel, especially in regard to the elimination of pipes.

During all these tests it has been thoroughly proved that the principle of the process is right, but careful adherence to instructions is necessary in order to insure success. One of the most important factors is to introduce the anti-piping can at the right time. This time can be easily determined by noting the thickness of the crust which is formed on the surface of the ingot in cooling.

Dr. Canaris writes as follows :

"As the results obtained were favorable from the first a systematic development of the process was carried out. On the basis of a great number of experiments made with ingots, weighing from 1 ton up, the quantity of thermit sufficient for each weight of ingot was accurately determined. It was then determined experimentally at what time, after pouring the ingot, the thermit should be introduced, and when more metal should be added in order to obtain the most favorable result. From these experiments, the following directions for this purpose can be laid down.

"The pouring of the ingots is accomplished in the usual way, either by direct pouring from the top or by the more complicated bottom-pouring method. The filled moulds are allowed to rest until the solidification has so far advanced, that on the walls of the mould a crust of considerable thickness is formed. The re-

¹ *Stahl und Eisen*, vol. xxxii., No. 8, pp. 302 to 311 (Feb. 22, 1912); *Metalurgical and Chemical Engineering*, vol. x., No. 4, pp. 232 to 234 (Apr., 1912).

² *Stahl und Eisen*, vol. xxxii., No. 38, p. 1579 (Sept. 19, 1912).

quired thickness of the crust depends, of course, principally on the size of the ingot. Then the anti-piping thermit can is pushed to the very bottom of the mould and is held there until the thermit reaction has taken place. The reaction results in a sudden and strong seething motion in the iron. After the reaction ceases, the surface of the still liquid material sinks, according to its condition, from 70 to 150 mm (3 to 6 in.), while the solidified walls remain intact. Then enough hot metal is quickly added, by direct pouring from the ladle, to bring the surface of the still liquid interior of the ingot to the same level as the solidified walls. The moulds are then immediately covered and the ingots left to cool."

If the ingots are poured from the top there is more hot material on the surface and therefore the crust forms more slowly, whereas the interior of the ingots cools more quickly. For this reason the can must be introduced earlier than in the case of bottom pouring, as the steel nearer the bottom is cooler.

In top pouring, the right moment for the introduction of the can is when the crust formed has extended over about one-half the surface of the ingot. In an ingot of 5 tons it takes, under normal circumstances, about 7 min. for the cooling to proceed to the proper point for the introduction of the can.

Should it not be possible to add more metal, the hollow space formed by the sinking of the metal should be filled up immediately with fine, dry sand and covered in the usual way.

If the ingots are covered, and neither the additional pouring made nor the hollow space filled with sand, the steel will rise again to the cover. In this case, although segregation and large blow-holes may be eliminated to a certain extent, the desired greater density of the material will not be reached.

The adding of metal necessary to fill the space left after the thermit reaction can be done in the manner best suited to the facilities of a given plant, but in general several groups of ingots can be poured before bringing the ladle back to refill the first ingots. In order to make the cover tight care must be taken that no metal is spilled on the upper surface of the crust, and that the hollow space is filled only to the lower edge of the crust. Any space between the cover and the ingot-mold should be made air-tight with moist sand, to prevent the escape of any metal forced up in cooling.

The can should not be introduced too soon, as the separation of the gases takes place just before the solidification of the metal. If the introduction is not made at the proper time, this

fact will be clearly indicated, as there will be no sinking of the metal in the mold. If, however, the can is introduced at the proper time, as determined by repeated experiments, the sinking of the metal in the mold will occur every time.

There is no danger that any parts of the thermit slag will remain in the interior of the ingot. Many of the ingots treated with the anti-piping thermit have been rolled into sheets and plates, and any detrimental amount of slag in the steel would certainly have made an appearance in the finished product.

Besides, the alumino-thermic process has the special advantage that through it any desired changes can be made in the proportion of the charge.

Figs. 1 to 6 illustrate the different stages in the treatment of steel ingots with anti-piping thermit:

Fig. 1 represents a section of the mold filled in the usual way, either from the top or bottom. This mold stands until a certain cooling off of the steel has taken place. The degree of the cooling can be determined approximately by watching the formation of the crust on the surface.

Fig. 2 shows how far the crust should extend before the introduction of thermit, in order to get the best results. In the case of a 5-ton ingot it takes about 7 min. for the crust to form properly.

Fig. 3 shows the can introduced. The can is quickly pushed down through the steel to the bottom of the mold and kept there until the reaction is over. The resulting seething of the steel will last about 5 seconds.

Fig. 4 represents a cross-section through the mold after the reaction has taken place. The empty space which formed indicates that the steel has become denser.

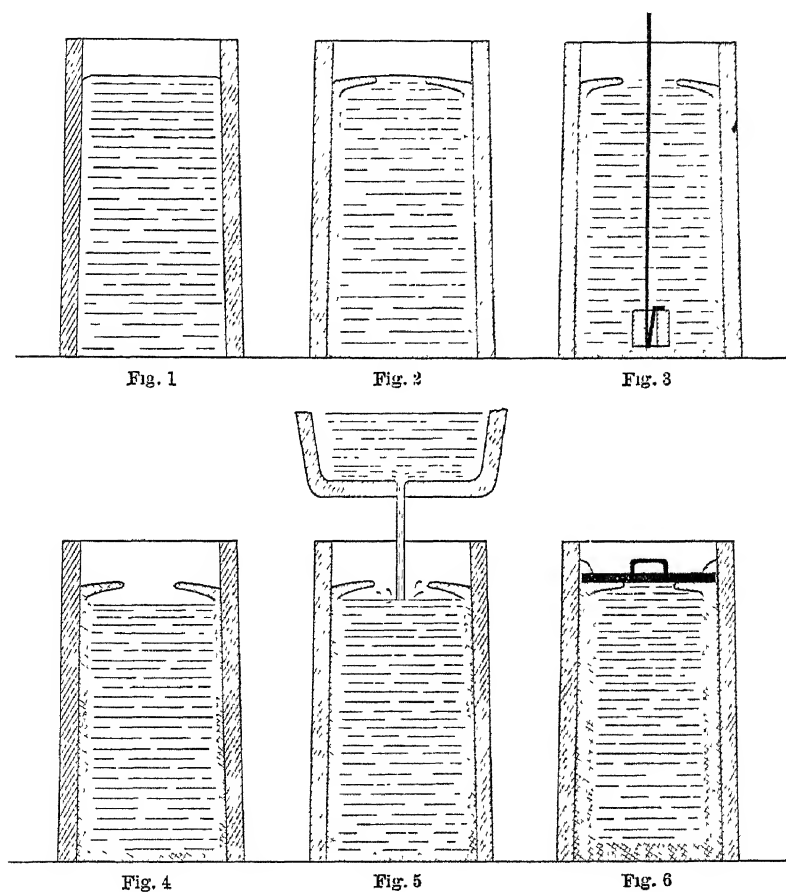
Fig. 5 illustrates the addition of new steel. Care must be taken that the added liquid steel does not rise over the edge of the solidified crust.

Fig. 6 shows the treated ingot after the lid has been properly put on.

Fig. 7 shows the shape of the can, and Fig. 8 the manner in which it is fastened on the iron rod. Scrap-iron rods from 0.5 to 0.75 in. in diameter, according to the size of the can, and in length about 2 ft. more than the height of the mold, should be used. Rods which are too thin have a tendency to bend, due

to the buoyancy of the anti-piping can. The reaction might then take place in the upper part of the ingot and the proper effect would not be obtained.

This process gives the desired results if the above-cited directions have been carefully followed. In most cases the second pouring can be arranged without difficulty, but if this should



FIGS. 1 TO 6.—DIFFERENT STAGES IN THE TREATMENT OF STEEL INGOTS WITH ANTI-PIPING THERMIT.

not be possible, the empty space between the solidified crust and the surface of the ingot should be filled with dry sand, as already mentioned, and the covers put on in the usual way.

Although some steel-works do not add the extra metal, claiming that a sufficiently improved quality of material is obtained without doing so, it should be done wherever possible,

since it adds materially to the amount of finished product per ingot.

Fig. 9 shows the lap in the plate rolled from steel treated with anti-piping thermit, and Fig. 10 one from untreated steel.

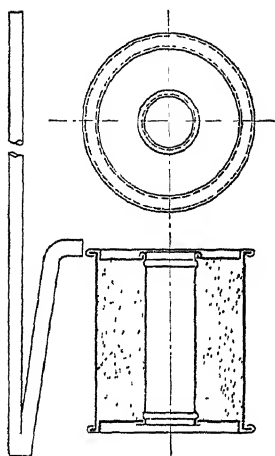


Fig. 7

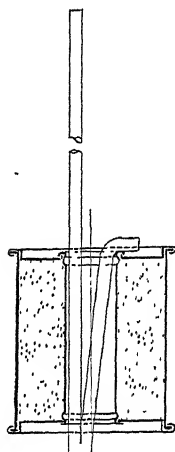


Fig. 8

FIGS. 7 AND 8.—CAN AND ROD FOR APPLICATION OF ANTI-PIPING THERMIT.

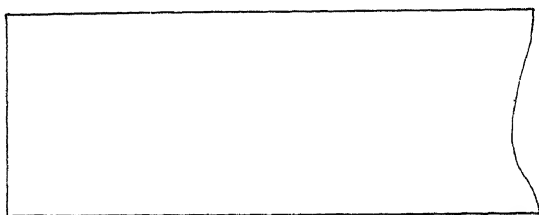


FIG. 9.—LAP IN PLATE ROLLED FROM TREATED STEEL.

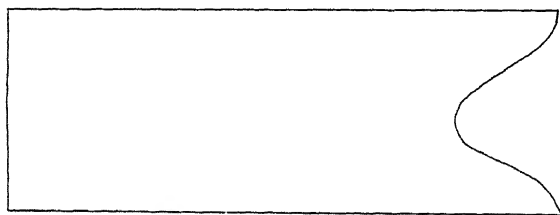


FIG. 10.—LAP IN PLATE ROLLED FROM UNTREATED STEEL.

The advantages derived from the use of anti-piping thermit in casting steel ingots may be divided into (I) improved quality of the material, and (II) better results of rolling.

I. *Quality*.—1. The steel becomes denser in proportion to amount of metal added after the thermit has been applied. Untreated ingots after solidification have the same volume without the added steel and therefore are not as dense as the treated ones. The increased density is caused by the expulsion of the occluded gases due to the ebullition following the thermit reaction.

2. The removal of the already formed segregations towards the middle of the ingot and their transfer to the surface of the ingot; further segregations are avoided through equalization of the temperature of the steel.

II. *Rolling*.—1. Elimination of the laps which occur from piped ingots, or from ingots with a sunken or irregular head.

2. Better physical properties of the steel in the heads.

3. A saving of from 3 to 10 per cent. of the finished material will be obtained. An average of 5 per cent. would give a profit of about 75 cents per ton.

4. Elimination of defective plates caused by large blow-holes on secondary pipes.

5. In small ingots the cost of the treatment amounts to about 35 cents per ton in the open-hearth department; against this there is an estimated saving of about 75 cents per ton in the rolling department. In large ingots (about 8 tons and more) the cost of the treatment is only 20 cents per ton and the saving is the same as in a small one.

6. A still further saving, which could not be calculated exactly, will be derived from the decreased percentage of defective plates, which is cut down to about 0.3 per cent., according to Dr. Canaris. In this case all plates which do not give the required dimensions or have not the desired physical properties in the head are classed as defective. Untreated ingots of the same group are often not uniform. Frequently there is a sound ingot, which will give good results without thermit treatment, and next to it there may be one with a large pipe or blow-hole. Uniformity is secured by the thermit treatment.

Tables I. and II. show the chemical and physical properties of plates treated with anti-piping thermit in comparison with untreated materials. These tables have been taken from the paper of Dr. Canaris.³

³ *Metallurgical and Chemical Engineering*, vol. x., No 4, pp. 233 to 234 (Apr., 1912).

TABLE I.—*Analysis and Tensile Tests of Thermit Plates, Showing Uniformity.*

Analysis								
Head					Base.			
Plate	C	Mn	P.	S	C	Mn.	P.	S
	Per Cent	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1	0.082	0.40	0.022	0.040	0.078	0.40	0.019	0.038
2	0.072	0.49	0.048	0.046	0.070	0.48	0.033	0.040
3	0.076	0.45	0.028	0.020	0.070	0.43	0.026	0.018
4	0.071	0.47	0.040	0.030	0.069	0.45	0.032	0.026
5	0.082	0.45	0.028	0.020	0.074	0.43	0.026	0.018

Corresponding Test-Pieces.				
Head.			Base.	
Plate.	Tensile Strength.	Elongation	Tensile Strength.	Elongation.
	Lb. Per Sq. In.	Per Cent	Lb. Per Sq. In.	Per Cent
1	53,893	28.0	51,476	30.5
2	53,609	29.5	51,476	32.0
3	54,320	27.5	51,192	31.5
4	52,329	28.5	49,912	32.0
5	54,889	26.5	51,334	31.0

TABLE II.—*Comparative Results.*

Remarks	Plate	Ingot Weight.	Width of Plate	Thickness of Plate	Length of Lap.
		Pounds.	Inches.	Inches.	Inches.
With thermit.....	1	2,827	94.20	28/64ths	6.29
With thermit.....	2	2,849	84.00	25/64ths	4.72
With thermit.....	3	2,849	94.20	28/64ths	4.72
With thermit.....	4	2,838	94.20	28/64ths	4.72
Without thermit....	5	2,739	84.60	25/64ths	30.81

Tensile Strength.	Elongation.	Place where Test Piece was Taken.
Lb Per Sq. In.	Per Cent.	
1. 52,400	32.0	Base.
55,100	28.5	Head.
2. 52,100	30.0	Base.
53,800	28.0	Head.
3. 51,400	32.0	Base.
54,300	29.5	Head.
4. 51,800	31.0	Base.
54,200	29.5	Head.
5. 52,300	28.0	Base.
59,600	20.0	Head.

All new processes require time and experiment to determine exact rules for their application. The method above described was found to be practical for several plants. Slight changes may be necessary to meet special conditions, and a test long enough to show a good average result should be made in all cases.

Commercial Production of Sound Steel Ingots.

BY EMIL GATHMANN, BALTIMORE, MD

(New York Meeting, February, 1913.)

INTRODUCTION.

IN presenting this paper I will attempt to answer certain questions proposed at this meeting and describe and illustrate methods of producing sound steel in an economical and hence commercial manner, which are adaptable to the production of practically all steel manufacturers, by readily effected rational changes in the methods of casting, cooling, and subsequent handling of the ingots.

In some of the high-grade mills of America, where it is essential to obtain perfectly sound billets or blooms, it is the practice to discard from 35 to 40 per cent. of the crop- or pipe-end, and even with this excessive discard, secondary pipe or shrinkage-cavities are frequently discovered during subsequent working of the blooms and billets into their various manufactured products. The usual discard of from 10 to 20 per cent. from the crop-end of the ingot, which is made by the present generally employed methods of form and solidifying of the castings, certainly does not give the slightest assurance of the saleable product being physically sound or chemically homogeneous. I am aware that in the past there has been and at the present time there still is a decided difference of expert opinion as to what constitutes physically sound steel.

Physically sound steel, as I would classify it, must be free from blow-holes as well as pipe. The line of demarcation between harmless and harmful blow-holes is exceedingly difficult to define. It is certainly the better and safer practice to eliminate blow-holes from the saleable portion of the ingots and to form a well-defined shrinkage-cavity or pipe at the upper crop-end of ingot. The higher grades of steel are all piping-steels. Is not their superior physical quality due to this fact as well as to their better chemical composition? Such

is the case, in my opinion. Steel has frequently been chemically excellent in the furnace or ladle but of very poor or indifferent quality in the ingot and in the subsequent manufactured shapes.

Steel free of blow-holes, "degasified," pipes to an excessive degree as it contracts from the liquid to the solid, and segregates along the vertical central axis of the ingot unless some method be employed to check this action; hence in the ordinary practice an excessive discard is necessary from crop-end of ingot in order to obtain sound saleable steel. Numerous processes have been proposed to obviate this excessive piping, which may be broadly divided into two groups, based respectively upon mechanical and thermal action. One group comprises various systems of compression of the metal in the liquid state in which the contraction of the metal is compensated for by external mechanical action. The second group comprises the different processes in which it is attempted to retard the cooling in the upper part of the ingot either by superposing a hollow block of refractory material on the top of the ingot, or by directly heating the free surface of the metal by a coke- or charcoal-grate, plumbago, or gas-burner, or even an electric heating means. In all the processes of the second group the isothermal lines of the freezing or solidifying metallic mass of the ingot are distorted towards the horizontal, it being the form of these lines which determines the shape of the pipe, as well as the position of the richest segregate or zone of impurities.

Various types of both of these groups are employed in certain high-grade steel-mills both here and abroad with more or less success, but they are mostly quite expensive and not applicable to the economical production of large and rapid tonnage. The principles employed by me, and now in daily use at some of the largest high-grade steel-plants in America, belong, broadly speaking, to the last group, though differing in essential details from any method heretofore in use. Various modifications of these details have been worked out and employed in open-hearth and crucible steel-practice.

It has been my experience that the freezing or solidifying of an ingot which has been practically deoxidized or, as the term is used, "killed" in the mold, depends entirely upon the shape

of the horizontal cross-section of the ingot at its various planes from top to bottom, and also upon the thickness and consequent heat-absorptive power of various parts of the mold walls. This will be best understood by referring to the experimental group of four ingots, each of same actual volume and weight, cast of crucible-steel from same heat and ladle. The ingot, Fig. 1, was cast in a mold having uniform thickness of walls from top to bottom; the section at the lower portion was widest and thus the slowest to cool. The oxidized pipe extended approximately 60 per cent. of the vertical length and mass of the ingot and could be traced within 10 per cent. of the bottom of said mass in length of ingot. This ingot was shaped similar to those used in ordinary open-hearth practice with big end down, but had the same cross-sectional area of metal at the upper as at the lower portion, this area being rectangular at the upper portion and approximately square at the lower portion. The factors are respectively 4 by 4 at the lower and 3 by $5\frac{1}{2}$ at the upper portion.

Fig. 2 shows an ingot of similar cross-sectional area at the top as at the bottom, viz., it being 4 in. square at both top and bottom. The oxidized pipe in this instance extended 33 per cent. of the vertical depth and volume. The ingot below the oxidized pipe was sound. This ingot was cast in a mold similar to that used in ordinary crucible-steel practice, which has a uniform thickness of walls from top to bottom.

Fig. 3 represents an ingot cast with big end up, the upper sectional area being 4 in. square and the lower sectional area 3 in. square. The thickness of the mold wall was uniform from top to bottom. The oxidized pipe was approximately 20 per cent., the steel below the pipe being perfectly sound. This type of mold conforms to the ordinary "big-end-up" practice, which has been used to only a limited extent in America, but considerably abroad.

Fig. 4 is the photograph of an ingot cast in accordance with my method, wherein the mold walls were heavy and accordingly chilled the ingot rapidly for about 85 per cent. of its vertical height and mass, the upper portion of the walls being made thin and less heat absorptive. The oxidized pipe in this ingot is about 10 per cent. of the vertical height and mass.

These ingots were all made in split molds, necessary by

reason of their cross-sectional shape. For open-hearth or solid molds this practice would necessarily have to be somewhat modified, as described later in this paper. Specifically, the new method employs a metallic mold constructed to accelerate the cooling of the lower or greater portion of the molten mass and teemed ingot (approximately from 80 to 85 per cent.) and retards the cooling of the uppermost portion of the ingot, thus causing the upper portion to remain liquid longer and to act as a feeder. The upper portion of the ingot does not actually remain liquid much longer than in the usual practice for similar cross-sectional area, but as the cooling of the lower portion is greatly hastened a differential in cooling is obtained, which is really what is to be desired. Similar results have been obtained by employing fire-brick, or heating of the upper part of the ingot by coke, charcoal or the like; but these methods usually offset any saving in a reduction of the crop-ends, by their cost and delay in application.

According to my methods, it is essential that there be considerable lag in the cooling of the upper portion of the ingot compared with the time of freezing of the lower portion, hence, the distance from the vertical central axis of ingot is made much less at the bottom of ingot and progressively increases towards the top of the ingot. In crucible-steel ingots, which are largely made in split molds, the actual sectional area of the top and bottom may remain practically constant throughout the length of the ingot, but the distance from the vertical longitudinal axis to the surface of the ingot progressively increases towards the top. In ordinary big-end-up practice, where sufficient taper or differential in distance from the vertical axis to the surface of the ingot is given, to accomplish any notable reduction in depth of piping, the actual cross-sectional area of lower part of the ingot is much less than that at the upper part, hence the depth of pipe is not the true index of the actual volume or weight of cropping necessary to obtain physically sound steel.

One of the advantages of my system or type of ingot is that it is possible to obtain a practically uniform cross-sectional area at top and bottom of ingot and still obtain the benefits of the big-end-up type of mold.

Lifting of segregation is generally conceded to follow the



FIG. 1.—INGOT CAST WITH BIG END DOWN. UNIFORM THICKNESS OF INGOT MOLD WALLS.

FIG. 2.—INGOT HAVING SAME CROSS-SECTION AT TOP AND BOTTOM AND UNIFORM THICKNESS OF MOLD WALLS.



FIG. 3.—INGOT CAST WITH BIG END UP.
UNIFORM THICKNESS OF MOLD
WALLS.

FIG. 4.—INGOT CAST BY GATHMANN
METHOD, VIZ.: BIG END UP AND
THIN MOLD WALLS AT UPPER
PORTION.

reduction of pipe, and where the pipe is lifted the steel below undoubtedly becomes more homogeneous and freer from segregation.

For open-hearth practice with the big-end-up ingot, one of the greatest difficulties has been to devise a method of stripping and handling the ingots. I have worked out a method which

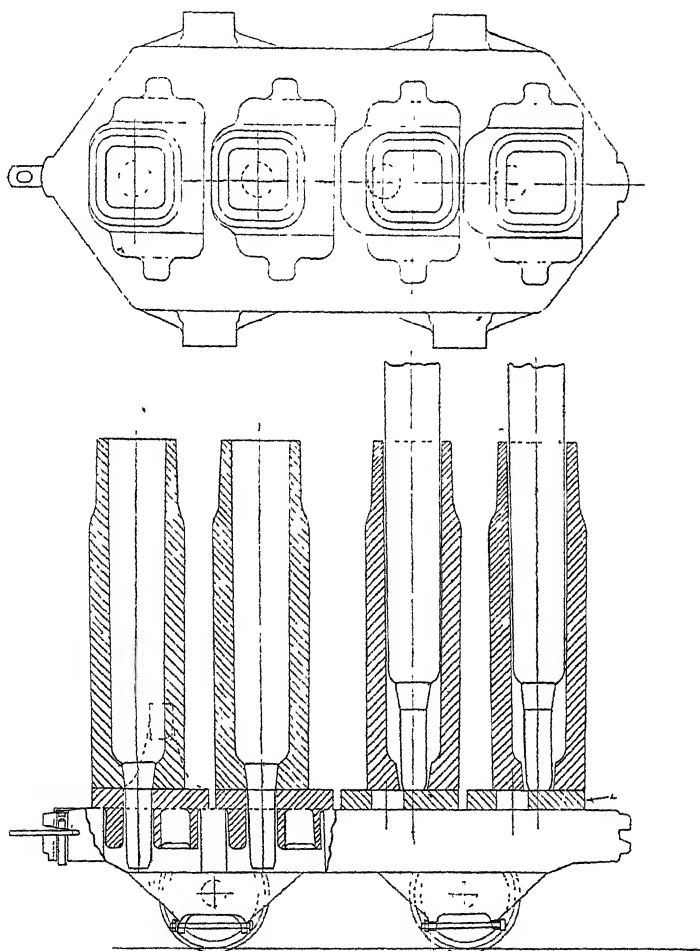


FIG. 5.—GATHMANN INGOT MOLD AND STOOL.

will accomplish this result and not interfere with the prevailing administrative practice or reduce the tonnage production; in fact, if anything, these new methods of stripping should increase the tonnage as well as insure sound steel with a greatly reduced cropping.

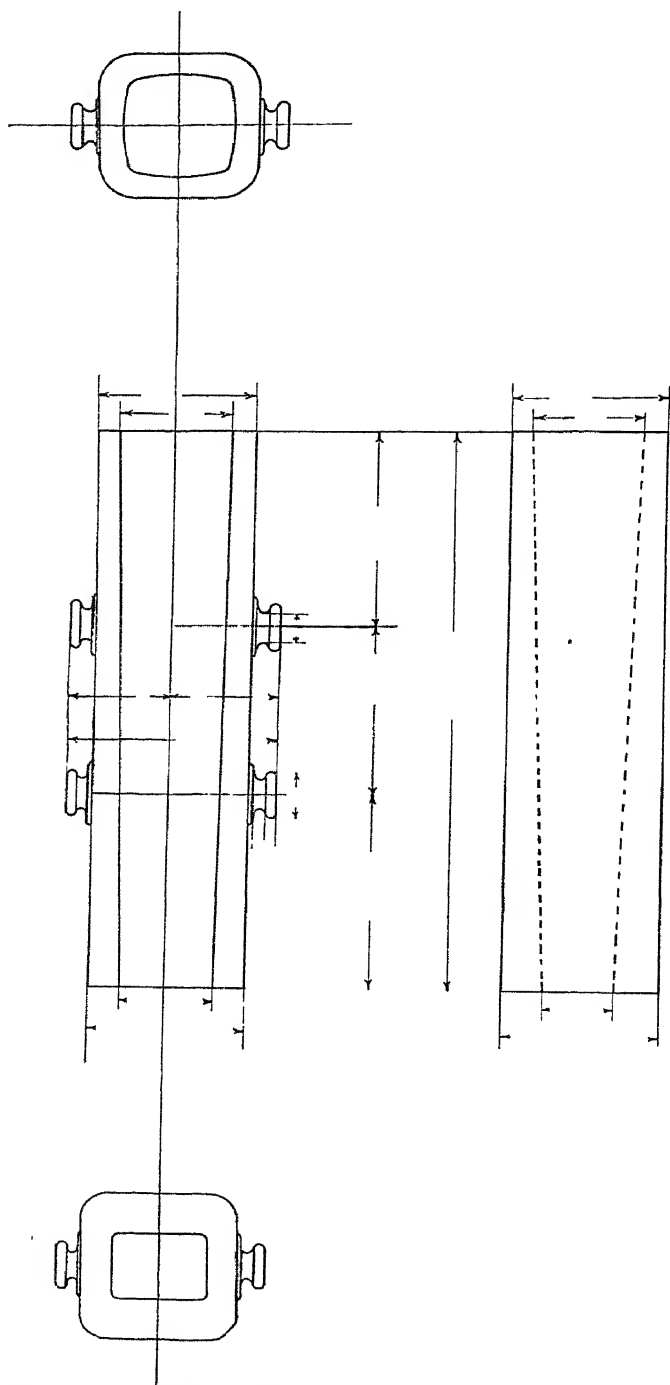


FIG. 6.—GATHMANN MOLD FOR BOTTOM CAST INGOTS.

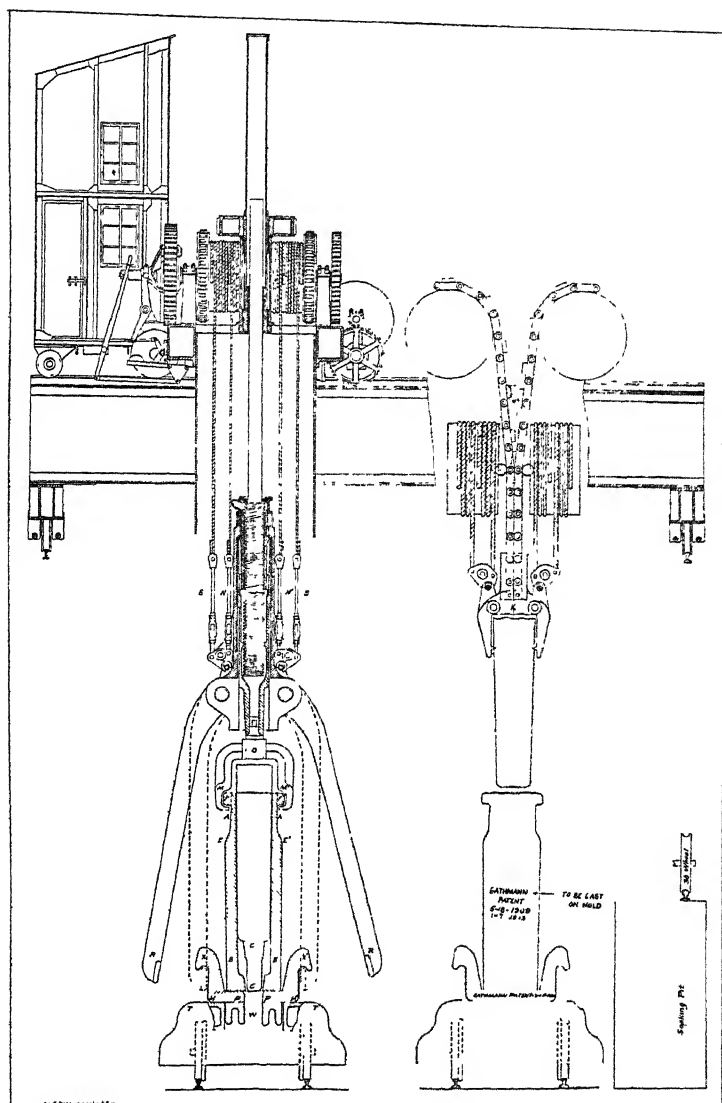


FIG. 7.—INGOT-STRIPPER AS MODIFIED FOR USE WITH GATHMANN MOLDS.

OPEN-HEARTH PRACTICE.

Fig. 5 shows the Gathmann ingot mold and stool on a car or buggy after teeming. The big-end-up mold will require a special type of stool to avoid such difficulties as might ordinarily be experienced in stripping, due to fins forming at the base of ingot and looking it to the mold. A downwardly tapered plug seals the base of the mold and projects through the stool. When the teeming is finished, the metal, by reason of the wedge-like taper at the lower portion of the ingot, is forced into virtual contact with the walls of the mold, which, due to the thickness of same, rapidly absorb the heat of the ingot. A slight differential or reverse taper of the ingot cavity is made from 15 to 20 per cent. from the top of the mold. The ingot in shrinking will automatically provide an air space at this portion, thus breaking the contact of ingot from mold walls and retarding the flow or loss of heat from this portion of ingot.

Fig. 6 shows in general elevation a modification of Gathmann mold without the stool. This type is used for bottom-cast ingots and ingots cast in pit-work. A special form of stool is required.

Fig. 7 shows a stripper of well-known construction with some details changed to suit the stripping of the big-end-up mold. The usual plunger is provided with a yoke whereby the lifting-lugs on the mold may be engaged. The plunger may then be raised or lowered, thus raising or lowering the mold, and in case of the ingot sticking in the mold upon lowering, the yoke is forced downwards on the mold, which rests on the combined sealing- and stripping-plug; the ingot is thus projected above the upper part of mold, or rather the mold is lowered down, exposing the upper part of ingot. If a "sticker" is encountered the weight of the thrust from yoke on the upper portion of mold is taken up by the hanging arms, $r-r'$, which engage the lugs, $x-x'$, on the stool, $p-p'$. The usual soaking-pit crane is employed to remove the ingot entirely after it is exposed above the mold walls, the car carrying the ingot contained in the mold being run into the proper position for this operation.

Fig. 8 shows a slab ingot mold designed for plate- or sheet-

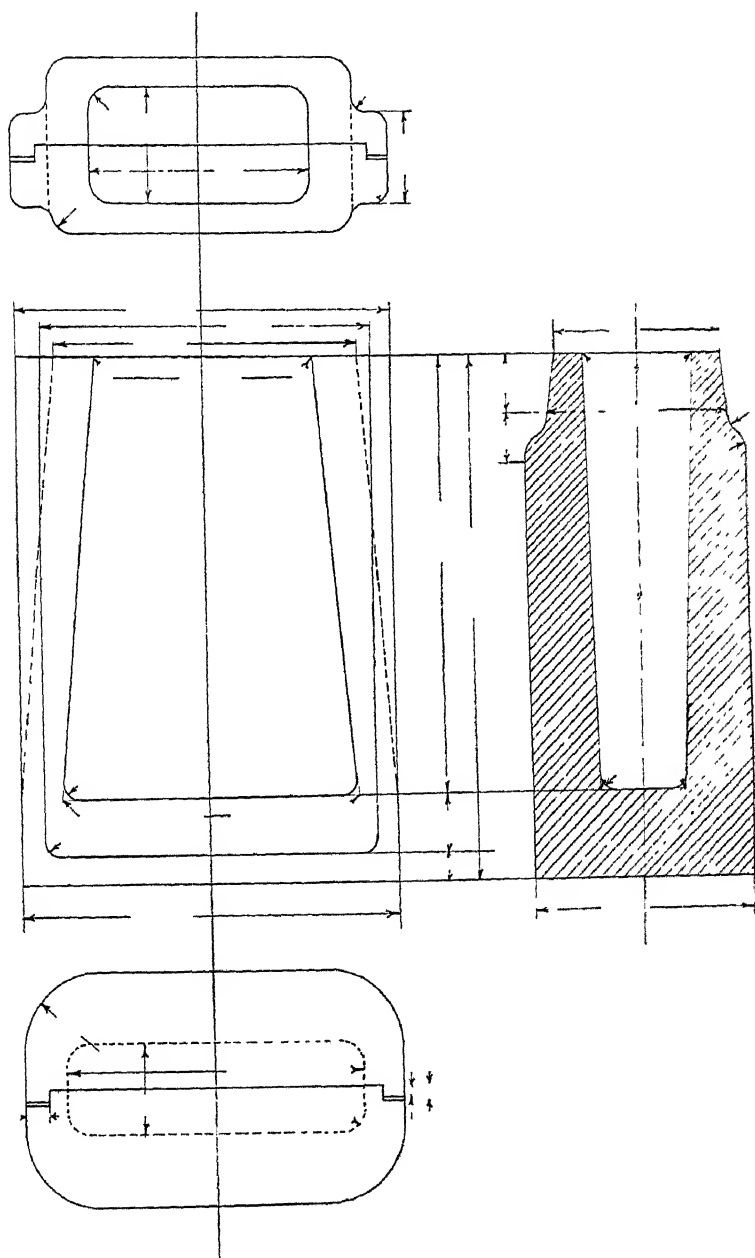


FIG. 8.—GATHMANN INGOT MOLD FOR SLAB INGOTS.

mills desiring to roll directly from the ingots. This ingot is made similar to the crucible-steel ingots, with uniform cross-area at top and bottom so as to obtain equal width from finished plates or sheets. The ingot may be designed of approximately the following general dimensions: 20 by 10 in. at the upper portion and 26 by 7 in. at the lower portion. The nearest distance from the vertical central axis of ingot to the surface of the same would thus be 5 in. at the top of the ingot, and but 3.5 in. at the lower part, with approximate progressive dimensions intermediate. This ingot is cast in a similar design of mold as previously described, wherein the cooling is greatly accelerated by means of absorptive action of heavy mold walls. This mold is necessarily made in two parts and held together with clamps in order to allow ready stripping of ingot.

RÉSUMÉ.

The general practice as outlined in the above brief description reduces the pipe in the deoxidized or killed steel so that with an average discard of 12 per cent. sound metal will be obtained. Segregation will undoubtedly be disposed of in direct ratio to the lifting of the pipe. The stripping apparatus and bottom-sealing stool make practicable the use of the big-end-up ingot for car practice without complications in removing the ingot.

I desire to repeat that this system is not in an experimental stage, as many tons of steel are daily being cast in both open-hearth and crucible plants in accordance therewith.

The features which require especial mention are:

- (1) No increase in cost of manufacture.
- (2) Simplicity in (a) construction and (b) operation.
- (3) No radical interference with present plant practice.
- (4) No skilled labor or supervision and attention required beyond such as may at present prevail in any mill.
- (5) (Most important of all.) It reduces the piping and provides physically sound steel by the accompanying lifting of segregation.

The methods, apparatus and ingot described in this paper are covered by United States and foreign letters patent.

Plant for Hadfield Method of Producing Sound Steel Ingots.

BY SIR ROBERT A. HADFIELD, HONORARY MEMBER, LONDON, ENGLAND.

(New York Meeting, February, 1913)

THE Hadfield method of producing sound steel ingots has been the subject of a paper¹ read before the Iron and Steel Institute, so that it will be unnecessary to describe it fully here.

The object of the method is to make possible the production of steel ingots in which the pipe is almost entirely obliterated, and it is applicable particularly to the production of sound ingots, from which good rails could be rolled.

The method in brief consists in casting the ingots on cars, usually with the small end down. Each ingot mold is provided with a sand top, rammed up in the well-known manner. After the mold is filled to the desired height with steel, a layer of slag about 0.5 in. thick is placed upon it and the balance of the mold is then filled with charcoal. Then, through suitable piping, an air blast is directed in numerous jets upon the charcoal, which is burned thereby, the combustion supplying additional heat to the top of the ingot, which helps to keep the top fluid and to retard its solidification, while the lower parts are rapidly losing heat by its transfer to the mold.

By one plan the ingots are cast in molds placed upon bogies, movable trucks, or platforms, in the position A, Fig. 1. The ingots are cast vertically on bogies, say four or more ingots on each bogie, as may be found most convenient, and the upper surface of the steel is maintained in a molten condition during the transit of the ingots from the casting pit to the stripper. This is effected by a series of air pipes, for example, three in number, with jets or holes on the under surface of such pipes, placed preferably horizontally, arranged in such a manner as to cause jets of air to impinge on the upper surface of the ingots. These holes or jets are made throughout the entire length of the overhead pipes, the length of such pipes being

¹ *Journal of the Iron and Steel Institute*, vol. lxxxvi. (1912, II.), pp. 11 to 39.

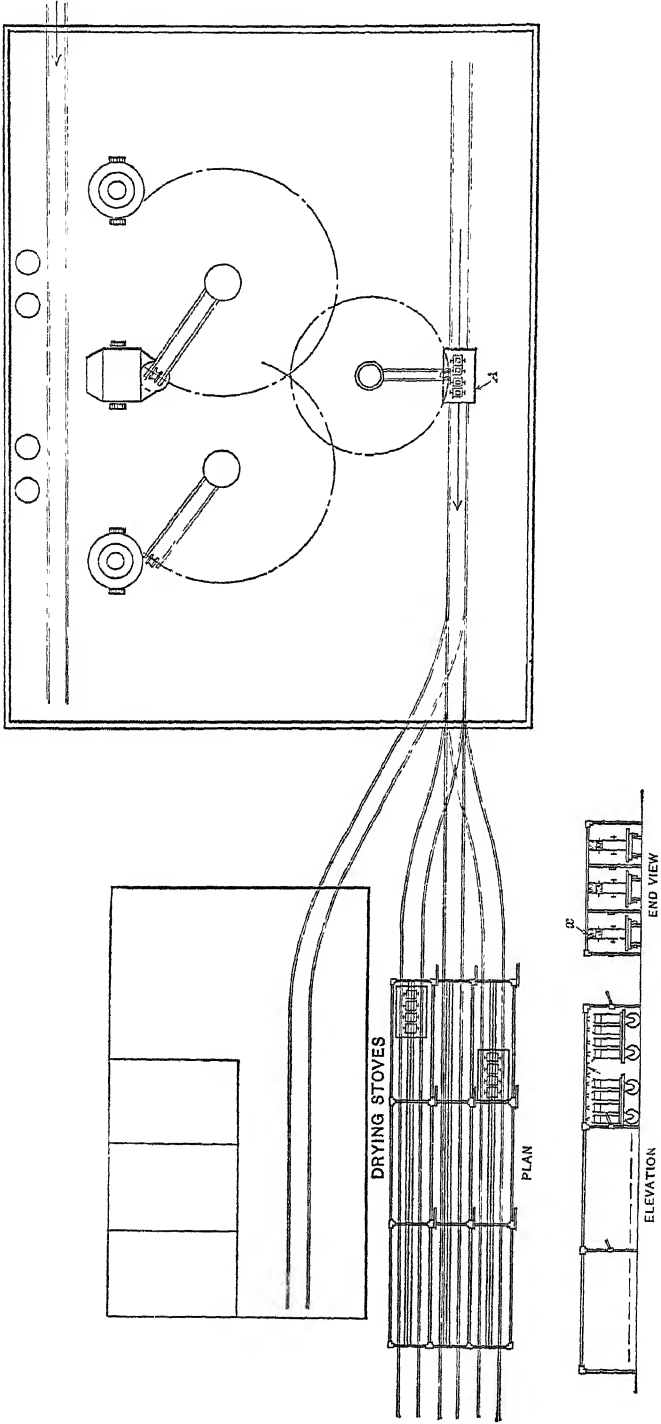


FIG. 1.—PLANT FOR HADFIELD METHOD OF PRODUCING SOUND STEEL INGOTS.

governed by the distance traversed under them by the bogies. In the figure shown the pipes are of sufficient length to extend over the tops of three rows of six bogies, each bogie containing four ingots.

One method of working is as follows :

As soon as possible after the ingots are cast, a layer of insulating slag and a sufficient quantity of charcoal for the purpose are placed on the top of each ingot. Charcoal itself delays to a certain extent the freezing of the molten steel, but a blast of air applied to the charcoal generates a considerable amount of heat in addition, in a manner which is impossible where such air blast is not used. The layer of slag, which may be termed an insulator, also prevents loss of heat and, to a large extent, prevents the charcoal from mixing with the steel, at any rate until too late to cause interference with the quality of the product below the head portion.

If a series of bogies extending the length of the overhead pipes is used, the air blast may be working during the whole length of the series, but, if it should happen that only one or two bogies are in transit under the pipes, the air may be shut off in sections, and only allowed to escape in the section in which the bogies or ingots happen to be.

Ingots may be cast in this way with either their large or small end at the bottom, whichever may be preferred, or practically parallel ingots may be produced of any desired size, section or shape.

This method may be also applied to the production of castings of various types.

Another modification to effect the same object consists in casting the ingots in molds upon bogies to which are attached electrically worked fans or blowers, the required electrical energy being taken from overhead or underground cables. A blower is mounted under the car at the side, or in such a convenient manner as may best suit the required purpose (see Fig. 2). As soon as the ingots are cast, the blower is put into operation, either while the car is standing still or as it moves along from the casting house. In this method a suitable supply of blast is obtained immediately after casting for each individual ingot or for a group combined on one bogie or other suitable movable or stationary casting arrangement.

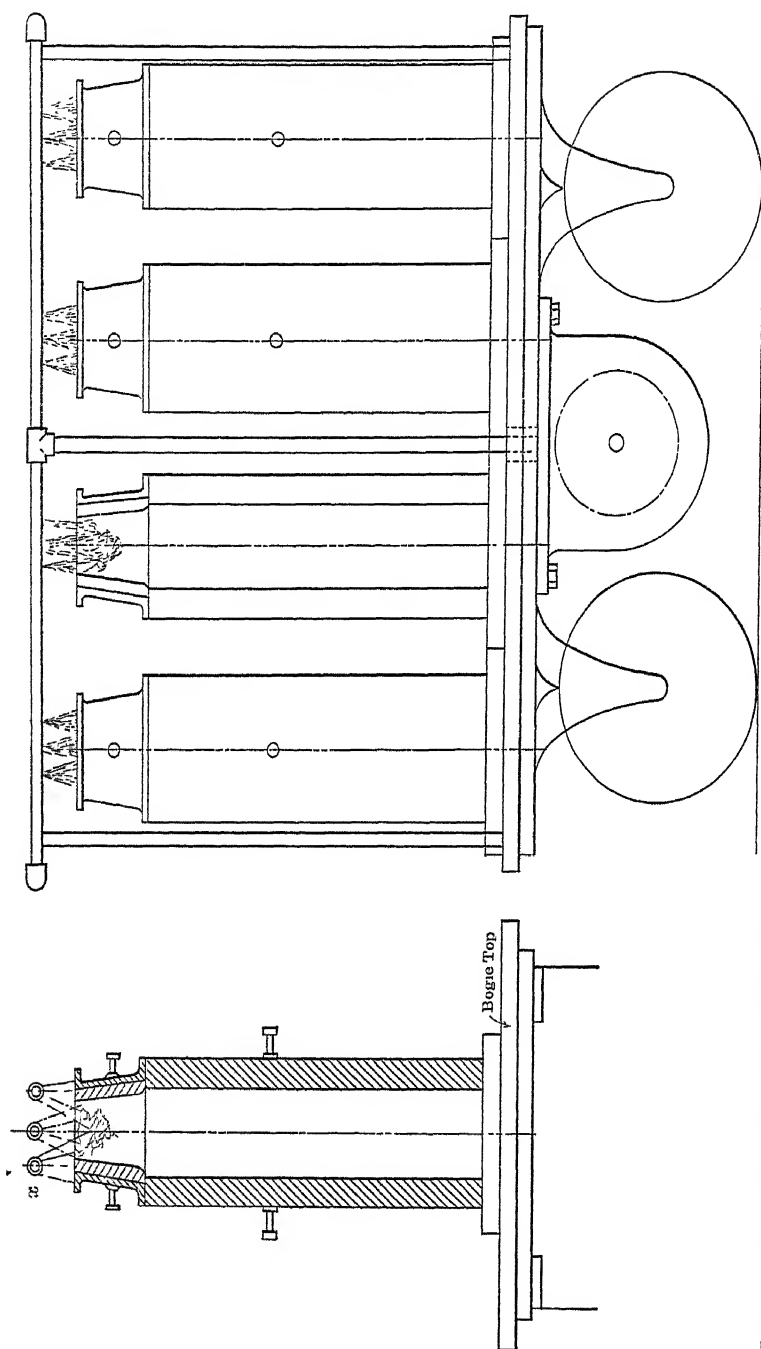


FIG. 2.—CAR PROVIDED WITH BLOWER AND PIPING FOR PRODUCING SOUND STEEL INGOTS BY THE HADFIELD METHOD.

The blast is continued as long as required; then the car or bogie is returned for further ingots to be cast.

To obtain a sound and safe rail, the steel must itself be sound and free from blow-holes. This requires the use of steel which will pipe or settle. As rail steel is usually made, the maker has no proper means of adequately dealing with this property of fluid steel, namely, the "piping." He therefore tries sometimes to avoid such steel, especially as it is dangerous unless the pipe is properly dealt with and fed, for the reason that in the methods in general use this pipe often runs down the center of the ingot for nearly one-half or even two-thirds of its length. By this system piping itself is avoided, proper feeding being effected. Thus sound ingots are obtained; that is, every one of 10,000 ingots can be made sound and free from piping, blow-holes, and segregation. In fact, 20,000 tons of sound ingots have been so made.

ESTIMATED COST OF PRODUCING SOUND STEEL INGOTS.

No. 1. Ordinary Steel.

Data upon which Estimates A, B, C, D, and E are Based.

Discard by Hadfield method of feeding.....	8 per cent.
Approximate cost of sand head, air blast, etc..	25 cents per ton.
Credit price of discard as scrap.....	\$15 per ton.
Cost of ingots.....	\$20 per ton.

A.—Cost of Ingots Made by the Author's Method Allowing 8 Per Cent. Discard.

100 tons of ingots at \$20 per ton.....	\$2,000
8 tons of scrap at \$15 per ton.....	120
92 tons of usable steel.....	1,880
Cost of sand heads, etc., at 25 cents per ton.....	25
	<hr/> 1,900
Or 100 tons of sound steel.....	2,070
Cost per ton of sound steel usable ingots.....	<hr/> \$20.70

B.—Cost of Ingots Made by the Ordinary Method Allowing 15 Per Cent. Discard.

100 tons of ingots at \$20 per ton.....	\$2,000
15 tons of scrap at \$15 per ton.....	225
85 tons of usable ingots.....	1,775
Or 100 tons of usable ingots.....	<hr/> 2,100
Cost per ton of usable ingots.....	<hr/> \$21.00

C.—Cost of Ingots Made by the Ordinary Method Allowing 25 Per Cent. Discard.

100 tons of ingots at \$20 per ton	\$2,000
25 tons of scrap at \$15 per ton.....	375
<hr/> 75 tons of usable ingots... ..	<hr/> 1,600
Or 100 tons of usable ingots.....	2,200
Cost per ton of usable ingots.....	\$22.00

Comparison of Methods A, B, and C.

Method.	Discard.	Cost Per 100 Tons.	Saving of A over B and C Per 100 Tons.	Saving Per 5,000 Tons.	Cost Per Ton.	Saving Per 1,000,000 Tons
	Per Cent					
A. Hadfield	8	\$2,070	\$20.70
B. Ordinary.....	15	2,085	\$15	\$750	21	\$150,000
C. Ordinary.....	25	2,165	95	4,700	22	950,000

*No. 2. Special Ingots.**D.—Cost of Ingots Made by the Author's Method Allowing 10 Per Cent. Discard.*

100 tons of special alloy steel at, say, \$25.....	\$2,500
10 tons of scrap at, say, \$15	150
<hr/> 90 tons of usable ingots.. ..	<hr/> 2,350
Or 100 tons of usable ingots.	2,600
Cost per ton of sound steel usable ingots.....	\$26.00

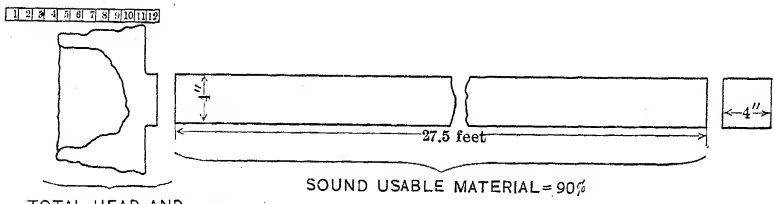
E.—Cost of Ingots Made by the Ordinary Method Allowing 20 Per Cent. Discard.

100 tons of special alloy steel ingots at, say, \$25.....	\$2,500
20 tons of scrap at, say, \$15.....	300
<hr/> 80 tons of usable ingots.....	<hr/> 2,200
Or 100 tons of usable ingots.....	2,750
Cost per ton of usable ingots.....	\$27.00

Comparison of Methods D and E.

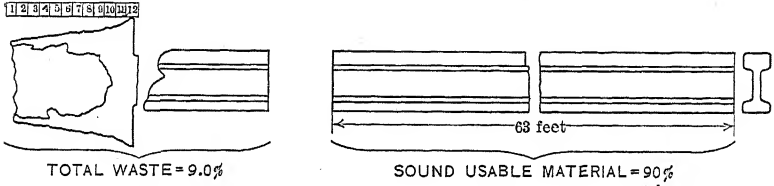
Method.	Discard.	Cost Per 100 Tons.	Saving of D over E Per 100 Tons.	Saving Per 5,000 Tons	Cost Per Ton.	Saving Per 1,000,000 Tons.
	Per Cent.					
D. Hadfield.....	10	\$2,600	\$26
E. Ordinary.....	20	2,750	\$140	\$7,000	27	\$1,400,000

NOTE.—Not only is the steel produced more cheaply by the Hadfield method, but the whole of the material is sound and free from piping, segregations, or other defects.



TOTAL HEAD AND
BILLET WASTE = 7½%

Total weight of ingot,	1,680 lb. = 100 per cent.
Weight of head and waste,	123 lb. = 7.5 per cent.
Scale,	2.5 per cent.
Sound usable billets,	1,513 lb. = 90 per cent.



TOTAL WASTE = 9.0%

Total weight of ingot,	2,296 lb.
Weight of head and waste,	214 lb.
Scrap,	9 per cent.
Scale	1 per cent.
Sound usable rail,	90 per cent. (Approx.)

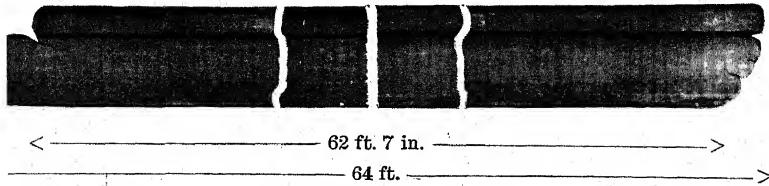
< ————— 12.25 in. ————— >

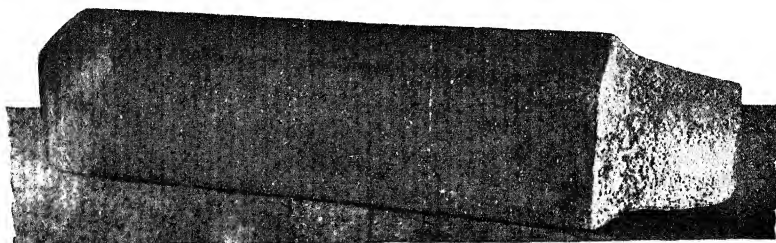


shed section
rom top of
ail near
ead.

Ingot head showing sound fracture.
Representing only 8 per cent. of
ingot.

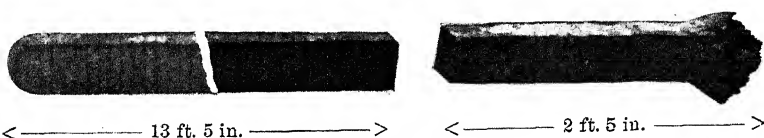
Rail discard
weighing 24
lb. Near
head.



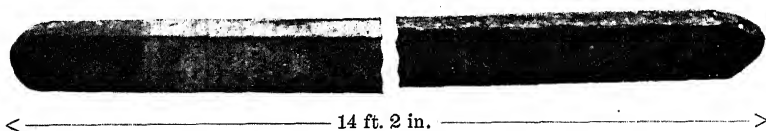


11-in. Ingot weighing 1,680 lb.

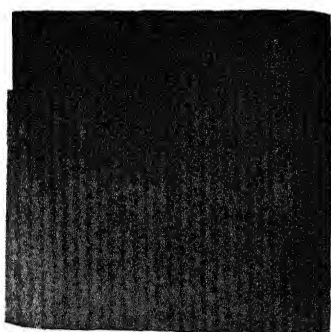
A. B.



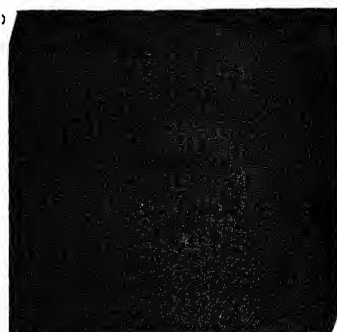
Billet from top portion of ingot.
Section removed between A and B, 1 in.



Billet from bottom portion of ingot.



Polished section from top portion
of billet at part marked A.



Polished section from top portion
of billet at part marked B, showing
last traces of piping.

HADFIELD PROCESS INGOT AND PRODUCTS ROLLED THEREFROM.

Sound Steel Ingots.

Discussion of the papers of Benjamin Talbot, p. 400 ; P. H. Dudley, p. 424 ; E. A. Beck, p. 453 ; Emil Gathmann, p. 461 ; and Sir Robert A. Hadfield, p. 473.

CHAIRMAN JAMES F. KEMP :—I call upon Prof. Albert Sauveur to open the general discussion on the subject of sound steel ingots.

ALBERT SAUVEUR, Cambridge, Mass. :—I believe that I have the privilege, this instant, of opening one of the most important, significant and far-reaching discussions ever held before this Institute. I believe that our talk this morning will be heard far and wide, not only on this side of the water, but also on the other side. Everywhere, in short, where steel is manufactured.

I welcome the opportunity of expressing my mind in regard to the production of sound steel ingots, a subject about which I entertain some very decided opinions.

I shall not attempt to discuss the relative merits of the various processes that have been described. I shall, on the contrary, confine myself to a general consideration of the important questions before us. So intensely do I feel regarding the production of sound steel ingots that I have put my remarks on paper in my desire to present my views.

I have always remembered my astonishment, not to say dismay, some twenty years ago, when I beheld for the first time a steel ingot, a rail steel ingot, that had been split open from top to bottom. There was the pipe ! There was an enormous cavity extending from a short distance from the top to nearly one-third of the length of the ingot ; and when I was told that manufacturers seldom discarded more than one-tenth of the total length of the ingot it became evident to me that a large proportion of piped, and therefore, defective, or at least weak and dangerous, rails were manufactured ; in fact, probably one rail out of six, nearly 17 per cent., of dangerous rails.

An investigation of broken rails extending over many years strengthened my belief that a large proportion of those failures were due to the presence of this hidden flaw, an unwelded or imperfectly welded pipe.

Year after year I called the attention of one of our technical societies to the objectionable practice, but my words fell upon unsympathetic ears. With few exceptions, both manufacturers and consumers contended that I was exaggerating the dangers of unsoundness—that the pipe was welded in the process of manufacture, that it was eliminated by the discard. Some, indeed, went so far as to claim that it did not form in properly cast ingots; others, that it was actually beneficial, since it meant the absence of blow-holes. By some I was put down as a bore, by others as a crank, while to many others I became *persona non grata*.

You will, therefore, pardon me if it is with a peculiar sense of pleasure, not entirely devoid of a certain victorious feeling, that I contemplate the present situation, when both manufacturers and consumers are at last willing to meet this problem fairly and squarely—to admit the dangerous character of the practice of producing unsound ingots, and to welcome practical cures.

Five or six processes have been described, which their inventors claim will produce sound ingots at a cost which is not prohibitive nor excessive. If these gentlemen are right—or if only one of them is right—then there is no longer any excuse for a steel manufacturer to produce unsound ingots, nor for the steel consumer—especially a rail consumer—to buy steel, especially steel rails, which he knows has been manufactured from unsound steel ingots.

Gentlemen, when we realize that human life is at stake, we cannot fail to consider this subject with thoroughness and earnestness.

THE CHAIRMAN :—I call upon Robert Job.

ROBERT JOB, Montreal, Canada :—I am very glad to be able to be here to-day and to listen to the discussions and to the reading of the papers which has occurred, because from my own work and observation I realize the extreme importance of

this work. In connection with mill practice, and particularly in the rolling of rails, I suppose no one realizes more than I do, the necessity of having sound, strong, and serviceable steel in our railroad tracks.

I have not had the opportunity of reading the papers, except in the case of those which we have just heard, and consequently I have not had any opportunity of going into detail more than in a very general way. I have noted the statements which are made upon the blackboard in regard to the discussion, and a few thoughts have occurred to me in connection with these.

The first question is: Is the present method of getting rid of the pipe by cropping, a safe and reliable device for making sound steel?

It occurs to me to remark that, in my own observation, with good mill practice very little piping is present; with poor mill practice 50 per cent. of cropping would not get sound steel.

The second question is: Is it desirable to increase the amount of metal arbitrarily cropped off the top of an ingot, and, if so, what would be a fair excess price to pay for cropping off 20 per cent.? I would say, for reasons which I have stated above, most emphatically no, since with good mill practice we have little or no difficulty due to pipe. Instead of specifying a definite percentage of discard, the proportion should not be stated, but should properly be left to the mill. Otherwise, good mill practice would be penalized. Tests should be made in such manner that, if a pipe or any other injurious defect is present, it will be detected and the material rejected. In connection with the mill test, of course, the problem comes up at once: how are we to detect injurious pipe or other injurious conditions if they are present, and I may mention briefly some tests which have been carried out under our direction upon the Lehigh Valley railroad during a number of years in the past, and particularly within the last couple of years, in connection with making tests of every ingot in a heat of rail steel at the mill. The plan, briefly, was simply to number the ingots, as can be easily done, and then to take a test butt from the top of each ingot under the drop. If it passes the test, accept the rails of the ingot. If it fails, simply reject those three or four rails from that particular ingot. We have found, I may say, that with the normal mill practice the work can be carried out very suc-

cessfully without hindering the rolling or other operations of the mill in any way. If the mill practice is bad, of course there is difficulty, as there should be, but with normal mill practice, as I have said, we have found no difficulty whatever and no tendency to block the mill.

The third question is: Is it commercially practicable to make ingots without pipes or blow-holes, and what additional expense of manufacture would be justified to accomplish this result? I may say that, in my own observation, it is commercially practicable to make steel sufficiently free from pipe and blow-holes to give good service; that is to say, speaking particularly of rail steel.

I was very much interested in Dr. Dudley's remarks in connection with the need and desirability of thoroughly deoxidizing the steel, for I think that too much attention cannot be paid to this work. As a result of investigating rails which have been in track under widely different conditions of service one becomes familiar with many types of failures, and realizes the necessity of having steel which is thoroughly sound. I think also that every engineer and every mill man welcomes the work which has been done with the object of bringing about a more thorough deoxidation of the steel and increasing the soundness of the product.

The ductility test is an excellent one, since it gives a better idea as to the liability of steel to break under sudden impact, and for that reason it has been adopted very widely and has certainly proved its efficiency.

There are very many problems of which one might speak at great length in connection with work of this importance, and with subjects such as those we have before us to-day, but in the brief time at our disposal it is hardly feasible.

THE CHAIRMAN:—The next speaker on this subject will be Max H. Wickhorst.

MAX H. WICKHORST, Chicago, Ill.:—The great bulk, say 95 per cent., of all rail failures, as they occur throughout the country, so far as their visible features are concerned, may be divided into three classes: First, we have the head failures, mostly crushed and split heads; that is, with an internal split or crack

occurring inside of the head, running longitudinally. Secondly, we have base failures; broken bases, moon-shaped or crescent breaks. Thirdly, failures which go by the name of "broken rails;" that is, where the break is through the whole section—what are called square breaks and angular breaks. The first type, head failures, can be traced to the interior condition of the ingots from which the rails were made, and they constitute, roughly, about 50 or 60 per cent. of the rail failures in this country. Now, that type of failure almost invariably shows a condition of segregation, and an etching will show more or less what may be called a spongy condition. That condition is to be traced back to the interior condition of the ingot, and that type of failure is confined almost entirely to the rails from the upper third of the ingot.

Now, as regards segregation, we must distinguish that very sharply from pipe. Segregation can be reduced by such de-oxidizing materials as silicon, titanium, and aluminum, and I presume that very likely manganese may have some effect that way, but the three materials that are valuable for reducing segregation are silicon, titanium, and aluminum. When these materials are used the concentrated segregation is avoided, but then the ingot also shows a deep pipe. The spongy condition, the small holes in the upper part of the ingot, can be eliminated, and that also seems to reduce segregation, but the result is a large central cavity. Now, this thing stands out prominently in the study of split head failures: that the origin of the flaw, the internal split, is in practically all cases clearly not a pipe that was in the ingot, but a condition of excessive segregation which rendered the material non-ductile in a transverse direction. Such excessive segregation in the interior is generally attended with sponginess of the upper metal and, what is also important, a softening of the metal in the outer part of the section around the segregated metal, so that the thing I want to call attention to is that apparently we do not have to be so much afraid of the pipe in itself, but of the heavy segregation.

If we can solve this question of piping and segregation, we have solved about 50 per cent. of the rail troubles. We have left still the other 50 per cent. of the rail failures, which I would

like to talk upon at some other time, possibly; but it hardly fits in with the present discussion.

THE CHAIRMAN :—The next speaker will be L. E. Howard, of the Simonds Manufacturing Co.

L. E. HOWARD, Lockport, N. Y. :—I regret very much indeed that we could not have made arrangements soon enough to have written a paper, and had some lantern slides to show what we are doing, but we only knew a few days before we came down here that we could have this opportunity, and the most we have been able to do is to bring two small ingots showing our process. These ingots will, no doubt, have some interest for you.

Our work is all in high-grade crucible steel, and in this the process has proved successful. In a brief way I will describe our process, which is of the lateral compression type. The compression unit is a box form of hydraulic press, the hydraulic cylinders forming one end of the box, and a plain steel casting forming the other end, with cast steel sides connecting these two members, and with tie rods, etc., holding the whole rigidly together. This press carries compressing dies, which may be of any shape to suit the ingots; immediately above these pressing dies is located a mold-carrying device, the molds themselves in the smaller sizes being of the split, or two-part, type. The press mechanism also includes means for lowering ingots from the molds, after they are poured, into a position between the pressing dies where they may be operated on. The ingots are stripped almost as soon as they are poured—which means, in the case of the split molds, opening up the molds by the mechanism provided almost as soon as the steel is poured, and just as soon as it set enough to hold together. The ingots are then lowered by hydraulic pressure into the pressing dies, and this is done so soon after pouring that if the ingots were handled at all roughly, either in taking them from the mold or in transferring them from the mold to the dies, they would burst open. The most that we have done is to provide apparatus of such a nature that we get the ingots from the molds and then into the compressing dies while they are still so hot that compression starts as soon as practicable after they are poured.

We started in with our first experiment on 15-lb. ingots, next made 200-lb. ingots, then 400- and 600-lb., and we are now designing an apparatus for 1,500- and 3,000-lb. ingots. As I said a moment ago, on ingots of this size—that is, probably up to 1 ton—we would use split molds. We are, however, working out a modification of the process to use the ordinary solid molds, such as are commonly used on larger ingots as in rail and structural mills, and this without departing from the general form of apparatus or method.

I was struck by one of the questions which was asked, and that is question No. 3; “Is it commercially practicable to make ingots without pipe or blow-holes, and what additional expense of manufacture would be justified to accomplish this result?” And I can say that in our own practice, which, as I said a moment ago, has been in connection with crucible steel only, it has been entirely a commercial success. In a general way, so far as we have gone, the cost of producing these ingots, including interest on the investment and all experimental work done up to this time, is slightly less than 3 per cent. of the value of the ingots. That may be misleading from your standpoint of rail steel, etc., on account of the different ingot costs, but from our results we think that something may be worked out which will be of interest to you.

I am sorry we could not have gone into this more fully, but we hope to publish before long a paper descriptive of the process more in detail, or an article in some of the technical journals.

THE CHAIRMAN:—The next speaker will be J. E. Sague, of the Public Service Commission.

J. E. SAGUE, Albany, N. Y.:—I did not come here with any purpose of discussing the papers, not having had an opportunity to read them. I have, however, been very much interested in the rail failure problem on account of having been Chairman of the Committee of the National Association of Railroad Commissioners which made a report on this subject after considerable inquiry. In that inquiry we searched the records of the various societies, especially of this society, and the records of the Railway Engineering Association, but, be-

yond that, we were helped greatly by personal conference with leading steel makers and with railway men, especially with such men as Dr. P. H. Dudley, A. W. Gibbs, of the Pennsylvania railroad, A. A. Stevenson, M. H. Wickhorst, and others who were willing to sit down quietly and tell us what they think of the rail problem. There is a good deal that these gentlemen think that they are not willing, apparently, to put into print. Some are willing and others are not. Well, in that way, I think we got some idea of the limits of the rail failure question.

Then we went to the reports of the Interstate Commerce Commission, and tried to find out what proportion of accidents are caused by the failure of rails, and there we found matter which indicated that the failure of rails has been much exaggerated from the accident standpoint; that is to say, the problem is not nearly as serious as we had felt it to be, judging by newspaper comments. But, nevertheless, it is sufficiently grave to warrant a great deal of careful investigation and, to some extent, anxiety.

We did not try to reach any final conclusions as to how rail manufacture could be improved. We are not experts. We felt that it was primarily incumbent upon us to indicate to the Railroad Commissioners throughout the country, and through them to the public perhaps, what the limits of the problem are; whether rail weights should be increased; whether a change should be made from the Bessemer to the open-hearth process; to what extent the steel mills are responsible; and to what extent the railroads are responsible because they have been increasing wheel weights enormously and increasing speed.

As a result, we made a report and gave a summary of our conclusions, and one of the principal conclusions was that a great deal more knowledge is necessary on this subject. Therefore I feel that I can express the views of most Railroad Commissioners in stating the great pleasure, the great profit, which will come to the country at large, to the railroads, and to the users of the railroads, from such careful studies as are being made here of rail ingots, because our investigations clearly showed that the start must be made at the ingot. It is entirely useless, I think, as indicated in the discussion here, to

expect sound rails unless we start with sound ingots. And to that end, every step of the process should be most carefully investigated.

But next to the rail maker comes the rail user, the railroad and the locomotive builder, and it is clearly indicated in the investigations which have been made, that a condition of extreme severity has been reached in the use of steel rails. The wheel weights have been increased to a point which would have been thought impossible a few years ago, and they still appear to be increasing. The hardness of the rail has greatly increased, until such men as President Farrell, of the U. S. Steel Corporation, and President Clarke, of the Lackawanna Steel Co., declare that we must go back to the manufacture of softer rails. Our committee did not see any positive reason why we should make a recommendation to that effect; but the question is one deserving of careful study.

Altogether, our investigation led us to conclude that the question is one which does not warrant despair or extreme pessimism. It indicated that a great deal of hope for the future is justified. We felt, in fact, that some increase in wheel weights might possibly be made as track gets better, as drainage is improved, as the depth of ballast is increased. We felt that the problem of rail breakage is influenced to a great extent by the question of speeds. It seemed to us that the railways, through their daily reports of rail failures, can determine fairly well how the speed factor influences rail breakage, and I think it is clearly shown, not only by our investigation, but by the studies of railroad men like Dr. Dudley, Mr. Gibbs, and others, that speed is one of the most important elements.

We concluded that the reduction of speed of such trains as the Twentieth Century Limited, the Pennsylvania Special, and similar trains, which was put into effect this winter, is of exceeding value in helping to solve the rail failure problem. We cannot expect to rush from New York to Chicago in 18 hours, through all sorts of weather, and over tracks that are heaved by frost, and count definitely upon getting to our destination safely. We have got to go a little slower, and these great railroads have done a public service in risking the criticism of some of their high-speed patrons and making this reduction, which will be followed by similar reductions of speed in winter

on other trains throughout the country. It is clear, we think, that what the public needs is a uniform schedule of trains on time, and not excessively high speed at particular times.

Now, gentlemen, I am very sorry that I did not know I was to be called upon, as I would have liked to make my discussion a great deal more comprehensive than it can possibly be under existing circumstances. I thank you.

THE CHAIRMAN:—The next speaker is James E. Howard, Engineer-Physicist of the U. S. Bureau of Standards.

JAMES E. HOWARD, Washington, D. C.:—A few observations of the condition of the metal in the ingot and in subsequent shapes in the blooming and rail mills have shown the early disappearance of evidences of blow-holes and central pipe as successive reductions take place. The observations were not sufficiently extended to permit of making a definite statement concerning the completeness of the welding of the walls of the interior cavities, but at least a fairly good welding of the steel was effected. Provided the welding was complete in certain places, then the structural state of the steel at such places would not be influenced by previously existing blow-holes.

It may be sufficient to say that, while a streaked appearance of the steel could generally be brought out upon etching, nevertheless the impression was gained that examples were presented in which effacement of ingot cavities was nearly or quite complete, and with such effacement the physical properties were doubtless brought to a very satisfactory condition. The elimination of blow-holes and ingot cavities of course remains desirable. Still, it is a question deserving consideration whether the attainment of a good structural state is not at times successfully reached in the finished shapes, notwithstanding that the condition of the metal of the ingot seemed unfavorable.

Work along this line was inaugurated several years ago, but was not brought to a state of completion. Attention was directed to the question of weldability and the definition of conditions which favored or precluded the welding of the steel.

The presence of slag inclusions, in globular form and in great numbers disseminated throughout the middle and lower parts of the ingot, was observed. These inclusions were present in

those parts of the ingot which remained after cropping. They were found in the lower two-thirds of the ingot, hence their removal is not a matter of proper discard.

From similarity in appearance of shapes rolled from the ingots in question with those from other ingots, it is inferred that a common cause for structural unsoundness is found in these inclusions. It may be definitely stated that welding cannot be effected across these slag globules or the filaments which result when they are extended in the rolled shapes. The question is not one of microstructure, which pertains to this feature of steel making, but of the elimination of an unweldable substance which causes streaks and laminations in the finished product, relatively unimportant in shapes which are stressed longitudinally, but which seriously detract from the value of the material when service conditions stress it transversely.

In size these slag inclusions range from a few hundredths of an inch in diameter down to a few thousandths of an inch. The lamination or streaked condition of the steel in consequence of such inclusions constitutes what seems to be one of the chief defects in steels for certain purposes. In the endeavor to produce sound steel it is clearly obvious that what constitutes unsoundness in commercial shapes should be recognized; and since the middle and lower parts of ingots are rolled into commercial shapes it is important that such parts should be structurally sound in the ingot or free from inclusions which preclude the attainment of sound steel in the finished product.

THE CHAIRMAN:—The next speaker will be W. C. Cushing, Chief Engineer of Maintenance of Way of the Pennsylvania Lines west of Pittsburgh.

W. C. CUSHING, Pittsburgh, Pa.:—I have nothing new to say, gentlemen, on this subject, other than what I have already said in the reports to the American Railway Engineering Association, in *Bulletin* No. 151. Sir Robert Hadfield's papers have also been reproduced in that *Bulletin*. I will call attention, though, to a statement taken from the report of the meeting of the Iron and Steel Institute of Great Britain last fall, which bears directly on the questions posted on the wall, which is, to

my mind, of great importance, and also is an answer to these questions. The following is taken from the paper of Dr. Puppe, of Germany, on Rolling Mill Practice, before the Iron and Steel Institute: "To standardize the amount of discard is beyond the mark, not only on account of the many conditions which influence segregation, but because every ingot requires individual treatment according to the depth of the pipe, which must at all cost be cut out. If the conditions of manufacture permit that price may be increased according to the percentage of discard, this may have the effect of a premium for the steel works which takes no interest in the improvement of its ingot or in the reduction of pipe. In German practice the elimination of piping is practicable and can be perfectly evolved. The end surfaces are milled, thus allowing end defects to become visible. Such defects are not revealed by any ordinary saw cut alone, as practiced in the United States."

The Rail Committee of the American Railway Engineering Association regards this method of testing as the best safeguard for the users of the railroads against the introduction of unsound rails. It is the best method which is at present before the railway engineers. That test is described in the speaker's paper as follows:

"The method of testing adopted, by which every test piece is broken (generally first nicked for facility of the work), is most valuable, for it enables the inspector to make examination for interior defects, which Dr. Puppe states the German engineer is able to make, except that our engineers do not yet require the parts to be milled or etched.

"When interior defects are found in the first test piece, the rails of that position in the ingot are rejected and the inspector passes to the next, or 'B,' rail. If this test piece also shows interior defects, he passes to the third set, or 'C,' rails, and if they, too, are defective, the rest of the heat is also rejected. In this way we have a graduated discard, which is valuable to the manufacturer as well as to the purchaser, for it prevents the waste of some good material which would be cut off by an arbitrary discard. The worst feature of a standard discard is, however, that the amount usually talked of is not deep enough to insure the removal of all the unsound material at some mills."

The last statement was further emphasized by Mr. Job in his remarks. I notice that Captain Hunt has given in his paper, which I obtained just before coming into the meeting, a very strong example of that difference in conditions at different mills. By reason of these variations the railway engineers consider that their plan, as now adopted, is the best at the present time; namely, the progressive discard upon examination by test.

Two or three gentlemen at the Iron and Steel Institute spoke of the rail failures in the United States in a very derogatory manner, and upon being given an opportunity for making some answer to them by Sir Robert Hadfield, I wrote the following:

"The statistics which have been kept show that the head failures are usually from 60 to 70 per cent. of the total failures, the web failures about 5 per cent., the base failures from 5 to 10 per cent., while broken rails are from 20 to 28 per cent. In special cases these percentages will vary considerably, and before the introduction of the heavy base sections the base failures were quite large."

This was written to illustrate the fact that European engineers have been confounding the term "breakage" with "defective rails." It is the practice in the United States to collect information about all rails which show defects, but, as already pointed out by Mr. Sague, large number of "rail failures" may be very misleading, because there may be little danger to the traffic, by reason of the warning given to the men by the nature of the failure. It is the broken rail which gives us the most worry and thought.

I will call attention to one thing which strikes me as needing some further explanation at present. Mr. Talbot, on the second page of his paper, says:

"Sound ingots, as regards the elimination of blow-holes, are produced by means of the well-known powerful deoxidizers, aluminum, silicon, and ferro-titanium. All these deoxidizers have the same effect when used in the necessary varying quantities to produce this. They all produce solid steel except for the large central cavity. They all diminish segregation. . . . The cost of the aluminum addition is very small; with silicon it is considerable, and with ferro-titanium it is large, to obtain the same result."

Dr. Dudley, in his paper, says :

“The suggestions of Mr. Talbot and Sir Robert Hadfield, to use a large percentage of aluminum in the ingots to reduce more completely the oxides, I do not consider advisable, from the difficulties already experienced with aluminum so used in rail steel for our heavy wheel-loads.”

As I understand the Goldschmidt thermit process, the use of aluminum is one of the essentials in connection with it. I may be wrong in this, but a note which I made some time ago explains that thermit is a mixture of finely divided aluminum and iron oxide which, when ignited, reacts to form superheated liquid steel and superheated liquid slag (aluminum oxide) at a temperature of $5,400^{\circ}$ F.

There is a flat contradiction in the recommendations, which seems to require clearing up on the part of those who are well posted on the reactions involved in connection with the use of aluminum, because if, as Mr. Talbot states, it is cheaper than ferro-titanium, there seems to be no use in using the more expensive ferro-titanium if aluminum will answer the purpose.

THE CHAIRMAN:—We will now hear from G. H. Clamer.

G. H. CLAMER, Philadelphia, Pa.:—The method here proposed is offered as a suggestion of a possible method of preventing pipes in ingots, by maintaining a molten top and allowing the solidification to proceed from the bottom upward.

To prevent pipes by keeping a molten top is by no means a new suggestion. This has been attempted in various ways, namely, by refractory plugs, hot slag, overflow of hot metal from one ingot to another, application of electrical heat, and more recently by the method of Hadfield of using a blast of air upon carbonaceous fuel.

The method here proposed is based upon the same principle upon which the Hering electric furnace is operated, namely, the production of heat in a body of liquid metal, the dimensions of which are so restricted that a current of high current-density flows through it, producing the “pinch effect,” which automatically expels the metal from the center of such restricted body of metal, causing it to flow in gradually around the periphery of the ejected core of liquid. This device is in

reality a furnace or heater of the resistance type, using liquid metal as the resistor.

I do not wish to take the time to go into the details of the operation of this device, but suffice it to say that experiments conducted in a small furnace of this type have been so satisfactory that four furnaces are now under way, ranging in capacity from 1 to 3 tons. It is the intention that the device here shown, which consists merely of a transformer directly connected with metallic electrodes and a block of magnesite in which the resistor holes are placed, shall be easily portable. It will, of course, be necessary to prepare a shield to protect the transformer from the heat given off from the cooling ingot. It has been calculated that transformers with capacity of from 10 to 50 kw. would take care of practically all commercial sizes of ingots. The current is transformed from one of 110, 220, or 440 volts to one of 4 to 6 volts, with a corresponding increase in the amperage.

To determine whether a device of this kind, or other device which contemplates the holding of the top of the ingot in a liquid condition, is commercially practicable, resolves itself into the question of cost and whether it can be used without interfering with the mill operation. These questions can only be determined by actual use under commercial conditions. The process is about to be tried out in this way, so at the present time no claims are made for it, and it is presented here simply as an undeveloped idea.

THE CHAIRMAN:—We will now call upon Dr. Allerton S. Cushman.

DR. ALLERTON S. CUSHMAN, Middletown, Ohio:—I did not expect to be invited to enter this discussion and did not come here prepared to speak to the question. My clients have been paying very close attention to this problem in their mill practice and are very much interested in it. We came here to-day to listen and learn from the experience of others who are also studying this very important subject. We intend to continue our experiments until we have assured ourselves that we are making sound ingots. The difficulties that have confronted us are unquestionably similar to those that other experimenters

have met with along this line. It is a very expensive and difficult sort of experimentation, and is complicated by the fact that one never knows what progress one is making until a number of experimental ingots have been split and thus opened up. There are, of course, a great number of variables that have to be studied and controlled before a consistently sound practice can be evolved. In the first place, the metal must be properly finished in the furnace and must be poured at the right temperature. Presumably there is an ideal pouring-temperature for every given metal, but I doubt very much whether even the most experienced melters could tell one what it is. Possibly this temperature will vary with weather conditions, depending upon the quantity of moisture in the air. This factor is of the greatest importance in blast-furnace practice, but, as far as I am aware, it is not taken into account in the operation of the open-hearth furnace. After we feel that we have a bath of metal properly finished, we have it in our power to vary: the conditions of pouring, including special treatments; the length of time the metal is held in the ladle; the teeming operation; special treatments in the ingot mold; the shape and size of the molds; the conditions and time of cooling; and finally the soaking pit and blooming mill operations. As I have stated, all these factors introduce a great number of variables into the problem, a change or mistake in regard to any one of which may be of very great importance. I cannot say at the present time that we have any contribution to make to this most important discussion. I can assure you, however, that if we had found anything which might be considered a contribution to the discussion I should feel inclined to most openly declare it. We are just now in process of installing a large ingot splitting device or saw which will enable us to economically split our largest sized ingots. As soon as this facility is at our disposal we shall probably be able to collect some data of value.

In the manufacture of the special material in which we are interested, our difficulty is probably somewhat different from that of the rail manufacturers, for the reason that our material does not show any tendency to pipe. Nearly all of our ingots at the present time are manufactured for rolling into sheet bars and sheets, and the tendency to pipe that some metals

show is not our principal difficulty. What we desire to attain is an absolutely sound ingot with all blow-holes eliminated, so as to avoid subsequent appearance of blisters or laminations in sheet bars or sheets. This problem is quite the opposite from the elimination of pipe, because, as has been well said here, any move made in the direction of a piping metal moves away from the tendency to produce open ingots from the standpoint of scattered or segregated blow-holes.

THE CHAIRMAN:—The next speaker I will call upon is N. Petinot, of the Titanium Alloy Manufacturing Co.

N. PETINOT, Niagara Falls, N. Y.:—In reading this very interesting paper of Dr. P. H. Dudley's I noticed on the carbon diagram of an ingot, Fig. 3, the following carbon determinations:

Line 9, a normal segregation of from 0.69 to 0.73,

Line 10, a negative segregation of from 0.69 to 0.59,

Line 12, a negative segregation of from 0.69 to 0.63,

Line 13, a negative segregation of from 0.68 to 0.65,

Line 14, a negative segregation of from 0.68 to 0.60.

I will endeavor to explain why this negative segregation occurs in the lower half of the ingot, and submit as a partial explanation the fact that the segregation is induced by the oxides in solution, upon which the carbon of the steel reacts.

In examining the silicon diagram of the same ingot, Fig. 7, it will be found that the silicon is nearly uniform. If we admit for a moment that there are some oxides in solution in the center of the ingot, and I will try to show later that it is proper that we should, why were these not reduced by the remaining silicon, which is supposed to be such an efficient deoxidizer?

This brings us to a discussion of the comparative efficiency of the three deoxidizers mentioned by Dr. P. H. Dudley and Mr. Talbot: viz., aluminum, silicon, and titanium.

Aluminum as a Deoxidizer.—Aluminum is the most powerful of all known deoxidizers. It will reduce the oxides of both silicon and titanium, but, unfortunately, by oxidation it will be converted into alumina, which melts at $2,010^{\circ}\text{C.}$, and con-

sequently is infusible at the temperature of the bath. This fact has received only slight consideration in many quarters.

This infusible alumina will remain in the steel, and if some part of it by chance is near the top of the ingot, it will cover the surface of the pipe and shrinkage cavity, eliminating any advantage which could be gained from immediate rolling, as suggested by Mr. Talbot. It will be noticed that Mr. Talbot has suggested in "well-made steel" an addition of 2 oz. of aluminum per ton, or in the case of a 12,000-lb. ingot an addition of 12 oz., which by oxidation will produce 24 oz. of alumina per ingot; this most infusible material will be disseminated throughout the mass in small shot, which later on will be rolled into streaks.

When such streaks of alumina are present between the metallic crystals of a steel rail they destroy the continuity of the metal. Having no strength in themselves, they are very readily broken, and cracks started from this cause are bound to spread through the adjacent metal when it is subjected to the severe stresses and strains of the heavy wheel loads passing over the rail.

Silicon as a Deoxidizer.—Silicon, as already mentioned, is not a powerful deoxidizer, and when added to steel it leaves a residue in it. Usual determinations of silicon give no clue as to the state in which it is found in the metal, whether as silicides, silicic acid, or silicates, but if in either of the two latter forms its presence is exceedingly harmful.

Microscopic examination of rail steel discloses small globules of foreign matter consisting of manganese sulphide and also of oxides and slags. It has been noticed that very often a steel containing as much as 0.10 per cent. of silicon in the form of iron silicide will be wild when poured, necessitating an extra addition of aluminum as a deoxidizer in the molds.

Titanium as a Deoxidizer.—Titanium added to the steel in the ladle, after the silicon, will complete the deoxidation. Heats with the addition of titanium of from 0.05 to 0.10 per cent. will not require any additional deoxidizer in the molds. It is the only low-priced deoxidizer known which can be used without danger of leaving any of the products of its oxidation in the bath of steel, as pointed out in Dr. P. H. Dudley's paper.

By oxidation titanium produces titanous acid, which will flux

the slag in suspension in the steel, and, if sufficient time be given after its addition for the reaction to be completed in the ladle, all of the products of its oxidation will rise to the surface, leaving a clean steel.

If such a deoxidizer be used in the Talbot process to make pipeless ingots, the pipe being clean will weld more readily when the ingot is compressed.

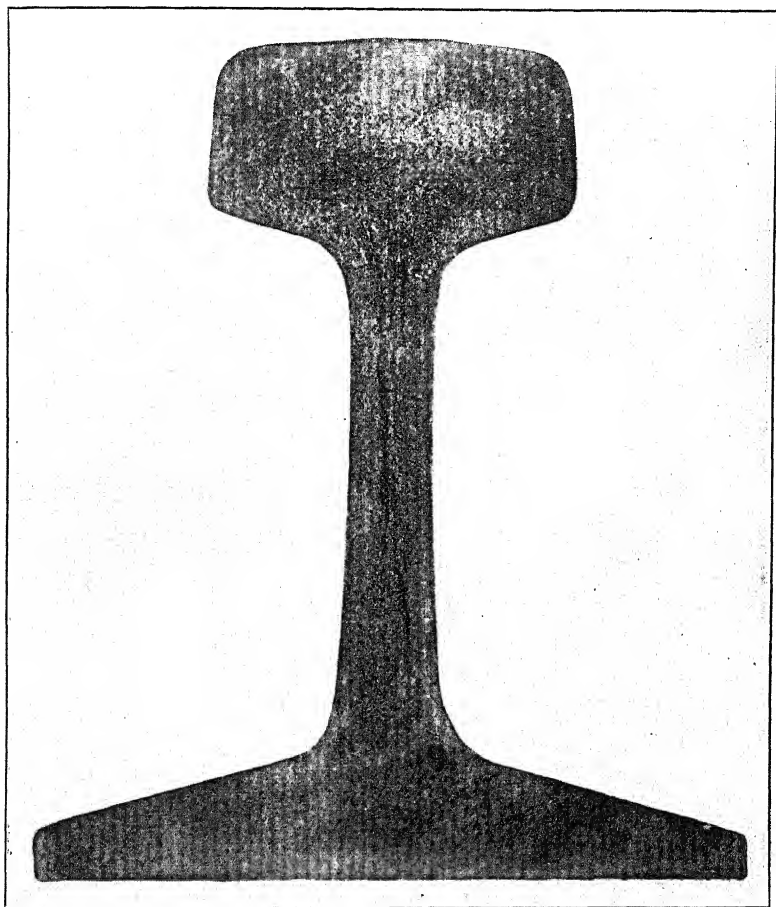


FIG. 1.—SULPHUR PRINT OF RAIL MADE OF OPEN-HEARTH STEEL
TREATED WITH TITANIUM.

If a comparison were made between two sulphur prints, Figs. 1 and 2, from the cross-section of two rails coming from the same part of two different ingots, the rails being of substantially the same chemical composition but the one having

been properly treated with 0.10 per cent. of metallic titanium, the other being a plain open-hearth steel, the former would not show any slag inclusion and only a very slight segregation, while the latter would in many instances show a dirty steel, full of slag, and an excessive amount of segregation.

It will be noticed that the untreated rail has a silicon content of 0.15 per cent., and it is a fact that a few pieces of

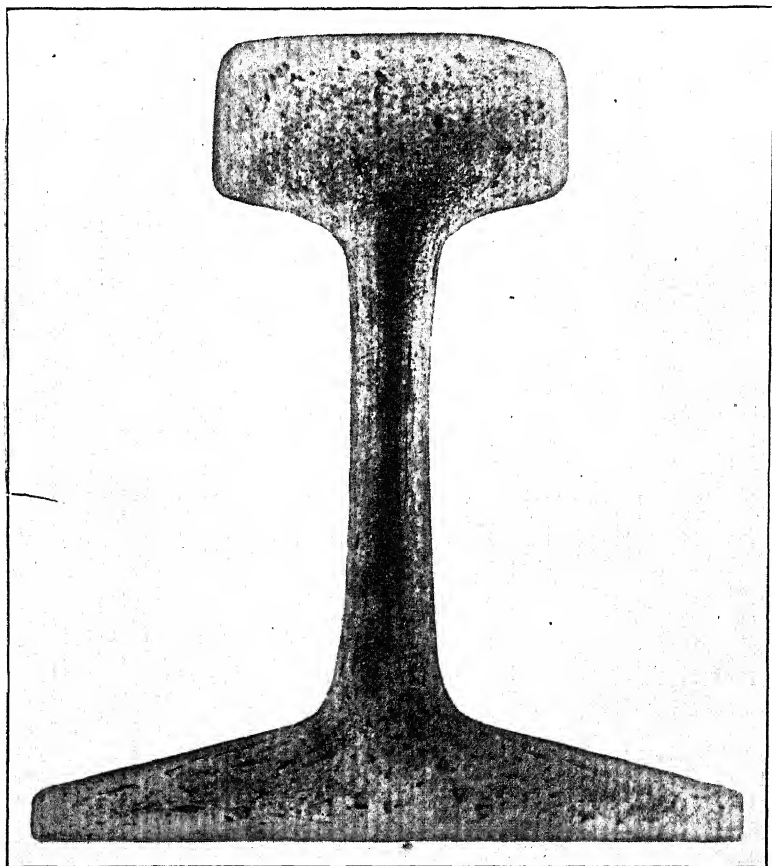


FIG. 2.—SULPHUR PRINT OF STANDARD OPEN-HEARTH STEEL RAIL.

aluminum were used in the mold in the production of this steel. If, therefore, silicon is a powerful deoxidizer, why should there be such an amount of oxides and slags, and consequently so much segregation in this steel?

The sulphur print of the rail treated with ferro carbon-titanium does not show anything of the kind and it is, therefore,

reasonable to assume that the elimination of the oxides will result in diminished segregation.

From the foregoing arguments, my conclusion is that silicon, a cheap deoxidizer, ought to be used to a limited extent for the manufacture of good rail steel, but that deoxidation should be completed with titanium, which, as mentioned by Dr. Dudley, will not only react on the oxides, fluxing the slag, etc., but will also reduce the iron silicates formed in the steel by the previous introduction of ferro-silicon.

As to the impression that rail steel treated with ferro carbon-titanium pipes, I will say that good steel always pipes, and in experimenting along this line I have found—although I cannot as yet give a positive explanation for the fact—that silicon and titanium are antagonistic, and that the silicon content in rail steel treated with titanium should be reduced within the limits of from 0.06 to 0.10 per cent., for if the silicon is so reduced it will be found that the shrinkage cavity will be similar to the one shown in Fig. 1 of Dr. Dudley's paper, and that the cavity will be clean and free from all silicates, slag, or alumina, so as to increase its welding properties when the ingots are immediately bloomed, as suggested in Dr. Dudley's and Mr. Talbot's processes.

The drop and exhausted-ductility test, for basic open-hearth steel rails, is excellent, and we are indebted to Dr. P. H. Dudley for its introduction. Very often, however, a rail rolled from a heat treated with aluminum, in accordance with the ordinary practice or by the Talbot or the Hadfield process, might pass the drop test to the entire satisfaction of both the railroad and the mill, but, as already explained when I mentioned aluminum as a deoxidizer, such a rail containing streaks of alumina, after a certain length of time in service will develop defects which are very often the real origin of broken rails and consequent wrecks.

I wish to state, however, that I entirely approve of Dr. P. H. Dudley's specification for the manufacture of basic open-hearth rails, but will add, as already explained, that if no aluminum be used (which transformed into alumina will coat both the shrinkage cavity and the pipe) the steel bloomed under its equalized initial heat will be rendered pipeless by the usual discard of from 8 to 10 per cent.

THE CHAIRMAN:—I am sure we will all welcome a few words from Professor Howe on this subject.

PROF. HENRY M. HOWE, Bedford Hills, N. Y.:—I think we might look with advantage at why the pipe forms at all. It forms because the metal sags down on account of the shrinkage of the interior after the outer crust of the ingot has taken its size and shape. That sagging down can be prevented by means of blow-holes; that is to say, by allowing a little gas to develop within the solidifying ingot. The evolution of that gas puffs the metal up just enough, if it is managed right, to make up for the sagging which would otherwise take place. Thus, by allowing the gas cavities to form, you stop the pipe, and that method is perfectly effective.

But it has three serious objections: In the first place, when the blow-holes form your metal cannot be perfectly quiet. It must be a rather lively metal, and in that case there will be serious segregation. Quiet metal means metal which solidifies extremely rapidly, and under conditions which do not lead to segregation, for reasons which one easily sees on examination.

The second objection is the difficulty of welding these blow-holes themselves. In the case of very low carbon steel, such as tube steel or boiler plate steel, I should doubt if this was a serious difficulty, because here those blow-holes do seem to weld up.

The third objection is that what Mr. Hibbard calls sonims; any solid non-metallic matter present, such as slag, is squeezed into those blow-holes by the pressure caused by the still later evolution of gas. If it is squeezed into the blow-holes it cannot get out, but it remains concentrated there, making permanent impure masses, which will cause streaks or other defects in the subsequent rolling.

These are serious objections to closing up the pipe by means of puffing up the metal with blow-holes.

Instead of preventing the pipe from forming by this means, you can proceed in either of two other ways, each of which avoids the formation of blow-holes and the three evils which attend their formation.

You can allow the pipe to form, and then close it up by mechanical compression of one kind or another. Mr. Talbot's

way is not only interesting and ingenious, but promising. Sir Robert Hadfield's way also should be effective. Or, instead of this, you can prevent the pipe from forming, not by puffing the metal up by means of blow-holes, but by causing the solidification of the upper part of the ingot to lag behind that of the lower, so that the supply of liquid metal above may suffice to run down into the incipient pipe below and close it up as fast as it starts to form.

The lagging of the solidification of the ingot top can be brought about in divers ways, of which one is to pour the metal, especially the latter part of each ingot, extremely slowly. I do not feel at all sure that mill-men have gone to the limit in causing this top-lag.

I should not be surprised if it were found practicable to reduce the pipe very much beyond its present dimensions, reduce it very greatly, perhaps reduce it to insignificant dimensions, by further causing the solidification of the top to lag behind the solidification of the bottom.

Ingots are ordinarily made with the small end up, which hastens the solidification of the top, and prevents it from lagging behind that of the bottom.

Where ingots are cast from the bottom the conditions are the very worst possible. Here the bottom of the ingot is kept hot by the freshly-poured metal. The ingot should be cast from the top, with the large end up, in order that the top may solidify as slowly as possible. All that Sir Robert Hadfield and others do by applying heat at the top, is to delay solidification at the top. You can do that by administrative methods.

It is possible that another way—and I speak with some hesitation—may be to taper the mold so that the bottom of the mold would be extremely thick and the top of the mold would be exceedingly thin, so that the great thick mass of metal at the bottom of the mold would suck the heat out of the lower part of the ingot, and hasten the solidification of the lower part, and the top of the mold being relatively thin, the solidification at the top of the ingot would be relatively slow.

It is for administrators to see whether it is a practicable thing further to retard the solidification of the top by special methods of pouring. Can you pour the last of the metal very slowly? Can you come back and repour into the ingot, of

course doing this in such a way as to remelt the joint which tends to form where the first pouring stopped? Can you fill the major part of the ingot, say three-quarters or four-fifths of it, and then come back some minutes later and put in the last? Where successive heats are practically identical in composition, as in Bessemer mills, is it practicable to come back with some of the next heat and fill up the top of the mold? It may be that the mill-men have thought all these questions out and have answered them fully, but perhaps they have not. We often find that matters which we assume have been puzzled out, as a matter of fact have not been. If they have not threshed this matter out, I think there is room for serious thought in this direction, without going to the expense of the very ingenious process of Mr. Talbot or Sir Robert Hadfield's modification of the old processes—putting in a little buffer of slag. Is it necessary to go to any of these expensive methods, or can we, by administrative steps, arrange the pouring in such a way as to cause a sufficient lag of the solidification of the top?

THE CHAIRMAN:—We will now call upon K. W. Zimmerschied, Metallurgist of the General Motors Co.

K. W. ZIMMERSCHIED, Detroit, Mich.:—The discussion so far has been limited almost entirely to the subject of rail steel. As important as pipes and their concomitant troubles are in this product, we venture the assertion that they are even more important to the automobile builder, since in the nature of the case the latter is limited by such considerations as weight, to closer factors of strength in the bars and sheets he uses.

Now, I may say that our experience has been that probably 50 per cent. or more of our troubles are due, not to irregular composition of the steel nor to irregular treatment, but to defects that remain in bars as relics from the ingots. And these defective structures fall into five classes, generally allied to the subject under discussion and increasing in importance from first to last: ingotism, seaminess, pipes, segregates, and non-metallic impurities. Pipes and seams are put into their respective positions because, while they are probably the most dangerous defects to find in a finished product, they are at the same time those most easily detected in the processes of manu-

facture—noticeably in forging; hence relatively few piped and seamed members actually get into cars.

This discussion is limited to pipe, and yet it is impossible to divorce this subject entirely from others which are as closely related to it as cause to effect. I hope at some future time the Institute may see fit to hold other such symposia on the subjects just mentioned, some of which are as far reaching as that of pipe.

With regard to the first question: Is the present method of getting rid of the pipe a safe and reliable one? I would venture the opinion that it may be answered "yes," provided the inspection is carefully carried out on suitable sections, which have been allowed to cool. I do not believe that it is possible to detect pipe on the cropped end of a small red hot billet, but our experience has been that we can find pretty decisive indications of pipe on the cropped ends of billets when they can be carefully examined in the cold. We have followed this closely and feel that we can catch at least all dangerously piped material, and, in addition, some of the more decidedly segregated steel. Such piped material as does get by is so slightly affected as to give no trouble later on, at least.

With regard to question 2: Is it desirable to increase the arbitrary cropping? I think it is the consensus of opinion at this meeting that a specified percentage of crop as a cure for piping troubles is not feasible.

The answer to question 3: Is it commercially practicable to make ingots without pipes and blow-holes? may be found in several steel plants every day. As a matter of fact, we are now buying considerable tonnages of steel made by sound ingot processes without having to pay any advance in price for such material. Any good process of this sort should pay for itself in the decreased tonnage of remelted material arising from cropped ends and rejected steel. It is, however, possible that, on the cheaper grades of steel, this balance could not be struck and some additional charge would have to be added in order to cover the increased cost of such processes.

On question 4: What is the best process for making sound steel ingots? Mr. Hibbard's and Professor Howe's remarks strike at the fundamentals. We agree that the most promising principle to work on is one in which the action of the curative

agency, whether that agency be the application of pressure, addition of heat, or abstraction of heat, is continuous from the time of casting until solidification; and further, that the most promising curative agency is either graduated application of heat to the top of the ingot, or graduated abstraction of it from the bottom, so that cooling progresses from the bottom up.

One other solution of the trouble presents itself: The main difficulty in eradicating pipe from bars at present comes from the impossibility of welding up the internal cavity by rolling. If we could so handle the ingot that this welding would not have to take place the trouble might be solved in that way, and one of the heroic measures that might be applied would be the splitting of each ingot after solidification, but before the metal had cooled past the plastic state. It is probable, of course, that the cost of such a process would be out of the question at the present time, especially in competition with other sound ingot processes. At the same time, the development of special high-powered machinery has been responsible for even more spectacular measures than this and we submit that such a solution is not altogether out of the question. It would put the unweldable surfaces of the internal cavity on the outside of the section to be rolled and would also reveal in every case the exact depth and nature of each pipe.

THE CHAIRMAN:—The Secretary of the Iron and Steel Committee has some questions to propound.

BRADLEY STOUGHTON, New York, N. Y.:—I want to ask Dr. Dudley, since his paper has been commented upon by two or three people, especially in regard to aluminum, whether he agrees with Mr. Petinot that no aluminum should be used at all in making steel for rails, and also whether the segregation is reduced by the use of any of these deoxidizers that have been mentioned, especially aluminum, titanium, and silicon?

DR. P. H. DUDLEY:—I do not wish to use any aluminum in any rail steel, and the segregation is very decidedly reduced with the use of ferro-titanium. It is not so marked with a high content of silicon; it is reduced. You will notice that in the paper, from the carbon diagram. It is very decidedly reduced when you add one-tenth of 1 per cent. of metallic titanium. That has been our experience in making many thousand tons of rails.

THE CHAIRMAN:—We have about ten minutes before the hour of adjournment will arrive. Would the authors who spoke first care to add any replies to the points brought out by the other speakers? Would any member present, not regularly scheduled on the program, care to take part in the discussion? The opportunity is quite open, and we would be pleased to hear from any who care to discuss the question.

HENRY HESS, Philadelphia, Pa.:—About two years ago, or a little more than that, I was induced to buy an alleged secret process for making steel castings, which turned out to be no secret at all, as I found out later. It was with disconcerting promptitude that I was confronted with the sound-casting problem, which enters also into this question of sound rails and the reduction of pipes or their prevention. I started some experimental work on that, and thought it would be a good plan to look into the interior of an ingot while it was cooling. One interested friend was kind enough to call my attention to a valuable paper by Professor Howe and Mr. Stoughton on the study of paraffine ingots. Instead of paraffine I used resin first, and so far have used only that. My ingot mold was a test tube about $1\frac{1}{8}$ in. in diameter and 6 in. long. I found I was able to secure in this test tube ingot a pipe of any shape, any character, at any place. I could make the pipe large or small, or eliminate it altogether, and I was able to put that pipe at the top or the bottom, at the center, or at the outside, where it does not look like a pipe.

The work which has been done by Sir Robert Hadfield and others in heating the top of the ingot was the line which promised most success. The location of the pipe was determined by the location of the last heat application. You could drive the pipe at will from one location to another, top to middle, to bottom, to side and even outside; enlarge it or reduce it; flatten it or elongate it; in fact, by local heating and cooling you had the pipe under absolute control. Altogether, the study was a most interesting one.

So well was I satisfied that I had that whole thing solved when I drove my pipe to the outside of the small resin ingot that I immediately proceeded to have steel ingots made. So sure was I that I had something remarkable that I arranged for a photographic record of every $\frac{1}{16}$ -in. layer exposed by suc-

cessive planing. My idea was that I would have this record to prove beyond dispute that there was absolutely no pipe. I was not present when the first layer was planed off, but I received a call over the telephone asking whether I wanted the rest of the photographs made, as the first step had disclosed a 6 in. deep hole. Experiments are still proceeding, but there will be a less enthusiastic jumping at conclusions. Also, other materials of crystalline structure and ingots of gradually increasing dimensions will be tried out before again producing steel ones. This very short recital is not intended as a serious contribution, but rather to add a little to the humorous side.

H. F. MILLER, JR. (communication to the Secretary *):—Hadfield's method of producing sound ingots is good, and he has approached the problem correctly in that he has supplied a head for the ingot which has a slower rate of cooling than the rest of the ingot. This is due not so much to the device for keeping the charcoal hot with a blast of air as it is to the non-conducting mold that he provides for the head of the ingot. In heavy castings it has long been the custom to put a layer of carbon on top of the sink head so that the head will feed the casting instead of chilling and forming a pipe. The heads shrink gradually, feeding the casting, and, when broken, show a solid fracture. The steel is kept in a fluid condition for hours, so that the writer thinks that in ingot practice the ingot would be solidified long before the head, even if the blast of air is not used but only covered with a layer of carbon preferably hot, such as coke shoveled directly from a fire. This method would do away with the necessity of providing for a blast of air, which complicates the means of producing many ingots per day.

Steel would be much better in quality if the heat was soaked for at least one hour before tapping. That is, after the last addition of ore has been made and the ore has done its work, at least one hour should elapse before tapping. This is true also when steel for rails is made by recarburizing with molten cast iron, etc. The molten metal should be added at least one hour before tapping. The objection to this practice is that the production will be decreased, but I do not know of the production of any steel of highest quality which has been worked fast.

* Received Feb. 8, 1913.

In regard to steel produced by the electric furnace, it seems reasonable to assume that, if as much care were taken to produce steel in the ordinary type of open-hearth furnace as is taken in the electric furnace, steel of as good quality could be produced. By this statement I mean that, if the steel were produced in the usual open-hearth furnace in the usual way so that the steel was of good analysis, and if the slag were then run off and an entirely new slag of burned lime added to form a new non-oxidizing coating, the steel would be under reducing conditions and the occluded gases would escape, leaving quiet steel.

The oxidizing conditions prevailing in the open-hearth furnace after the bath has become coated with the slag are due to the constant reactions of the carbon in the bath with reducible oxides in the slag, such as silica, P_2O_5 , etc. If these oxides were drawn off and a new slag made, there could be no reaction take place, since no oxygen would exist in contact with the bath, and the carbon in the steel would soon put the bath in a reducing condition.

JOHN E. STEAD, Middlesbrough, England (communication to the Secretary *):—Dr. Howe long ago described what occurs when steel freezes in an ingot. In substance he stated that, when the greater part of the metal has become solid, the crystallites bridge across from the internal walls of the inside of the ingots as they slowly solidify, and that this mass sinks or sags downward as the liquid in the central column of liquid contracts. I have proved repeatedly that such action does occur.

The mass that sags must be free to flow as a whole; it could not sag if it were rigid. The bases of the fir-tree crystallites are rooted in the nearly solid walls and the branches are more or less interlocked. It is this which prevents them from falling to any great extent to the bottom of the liquid column. If, however, an ingot is very large and is cooled very slowly when the central column is still liquid, it is almost certain that many of the fir-tree crystallites will break away from the sides and fall to the lower part of the ingot. It is only on this hypothesis that one can explain how the steel in the lower central axis of large steel ingots is purer than the average. The greater quantity of the crystallites, however, do not fall, so the central mass,

* Received Feb. 15, 1913.

liquid and solid, sinks most freely in the center of the ingot, and the bridges of fir-tree crystallites form parabolic inverted domes, as shown in Figs. 3 and 4.

On passing the steel through the rolls, when in this condition, the plastic mixture is forced up into the pipe, filling it completely. If the central column is nearly solid and if it has assumed a mass resembling felt saturated with liquid, only the residual liquid rich in sulphur and phosphorus and carbon would be squeezed out of the closely interwoven crystallites, and this liquid would fill the pipe and yield a mechanically produced segregate in the upper central axis. This kind of segregation is more common than generally known.

To conduct the process properly, the solidification must not be allowed to proceed before pressing the ingot until only a small portion remains liquid. When steel freezes it passes first to a mixture of part solid and still liquid, which, as a whole, can flow under pressure, just as paper-pulp flows in the paper mill. Secondly, it reaches a condition like paper-pulp when a great part of the liquid is removed from the fiber, so that it will not flow, but readily breaks up or tears if tension is applied. Lastly, when all the steel is frozen, the mass becomes rigid and resembles finished paper from which the last portion of water has been removed.

It will be obvious then that, in every ingot, at one time during solidification, there is a central column of substance like paper-pulp that can flow, and this is surrounded by an annular layer of steel almost completely solid, incapable of flowing, but which readily breaks up on attempting to extend it, and which is, of course, most tender where it approaches the inside pulp, and strongest where it joins the annular layer of solid steel.

When an ingot in this physical condition is passed through the rolls the outer solid layer will extend without breaking and the inner, pulpy mass will flow, but the intermediate, annular layer will fracture and form fissures terminating only when they reach the completely solid steel. The steel immediately in front of the fissures will contain a preponderating proportion of fir-tree crystallites and a small quantity of liquid rich in metalloids, and in this state resembles a felt filter.

When the fissures form under the extension of the ingot in the rolls, the pulp inside the central axis will press against the

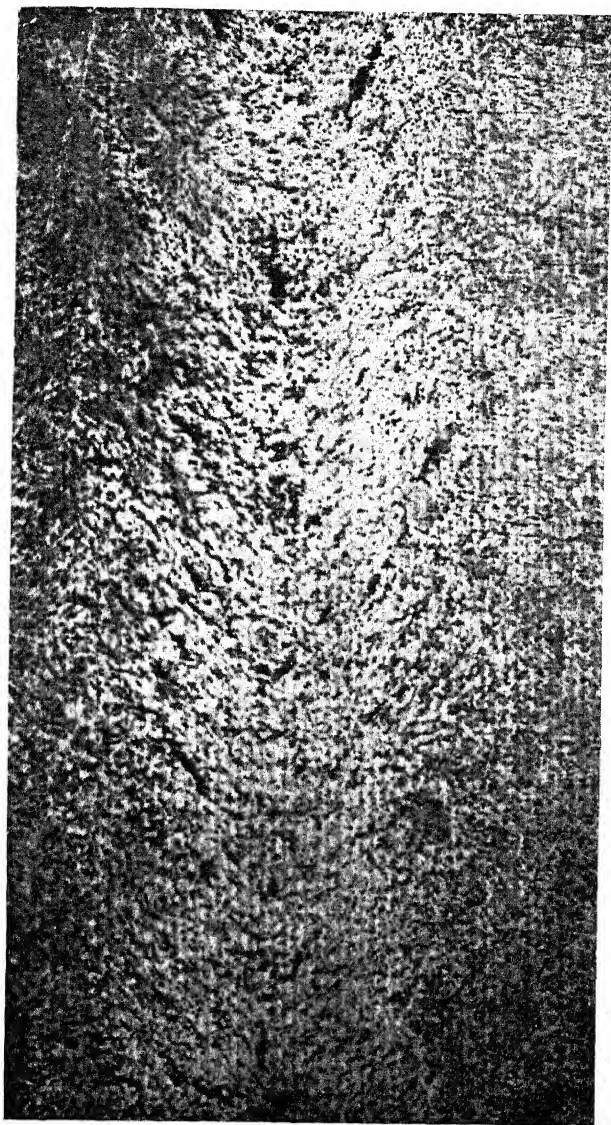


FIG. 3.—SULPHUR PRINT TAKEN FROM THE CENTRAL AXIS OF A 10-IN.
INGOT MADE QUITE SOLID BY ALUMINUM.

The wide end of the ingot mold was upward and the small end downward. The inverted domes could be traced from 6 in. from the top to 6 in. from the bottom of the ingot. It was cooled completely in air. The bridges were 2.5 to 3 in. across, about 30 per cent. of diameter.

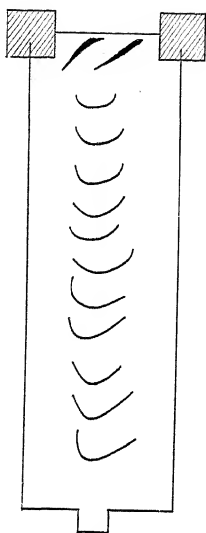


FIG. 4.—SKETCH OF INGOT FROM WHICH THE PRINT SHOWN IN FIG. 3 WAS TAKEN.

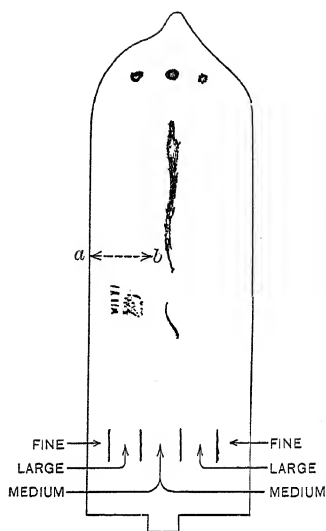


FIG. 6.—SKETCH OF INGOT FROM WHICH THE PRINT SHOWN IN FIG. 5 WAS TAKEN.

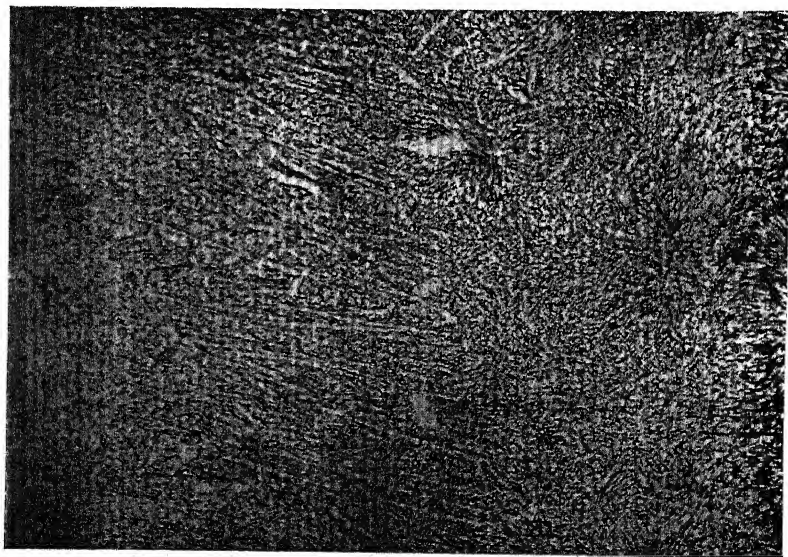


FIG. 5.—SULPHUR PRINT TAKEN FROM A VERTICAL SECTION OF A 10-IN. INGOT AT POINT *a-b* OF FIG. 6.

Outside layer 0.25 in.; crystallization very fine. The crystallites increase in length for about 2.5 to 3 in. The central axis is composed of finer crystallites: about 5 in.

felt-like filter and force the liquid rich in metalloids forward into the fissures, and this liquid will be followed by some of the liquid portion of the internal pulp, while the purer crystallites will be kept behind on the layer of the densely-packed crystallites, which constitute a real filter.

The examination of several auto-sulphur prints of complete ingot sections, and of blooms, has shown that the crystallites in 10-in. ingots—even without cooling in the soaking pit—are sometimes 0.2 in. and more in length. The remarkable feature is that the crystallites in the central column are generally smaller than those in the intermediate layers between the outside and the center. One explanation for this is based on the assumption that the crystallites shoot continuously into the central column during the time the mold is not disturbed, and that the smaller crystals form when the ingot is removed from the pit and the liquid inside is more or less shaken. A sulphur print, Fig. 5, taken from a vertical section of a 10-in. ingot at a point *a b*, Fig. 6, is here given.

PAUL KREUZPOINTNER, Altoona, Pa. (communication to the Secretary*):—To the experienced man the nature—that is, the extent and appearance—of a detail fracture of structural material, if it does break in detail, is an indication of the physical condition of that material, its homogeneity and purity. While the general appearance of all detail fractures is radically different from the appearance of sudden fractures, there is also a considerable variation in the appearance of detail fractures. If one has had the opportunity of everlastingly comparing fractures of all imaginable variations during a period covering well-nigh a generation, certain characteristics, indicative of quality, crop out, which, in connection with the results of other tests, permit certain general conclusions to be drawn which may be helpful in explaining some of the phenomena now under discussion. One of the deductions based upon my long experience in comparing detail fractures is that the character of this class of fractures has changed with increase in rapidity and mass-production of steel.

The percentage of area of the broken cross-section covered by a detail fracture has decreased and its appearance has

* Received Mar. 12, 1913.

changed. The sharpness in the individual lines separating each individual break has diminished considerably, and in many cases but little indication of individual breaks is visible. Still, the detail fracture as such retains its distinction from a sudden fracture.

How far are we wrong in our deduction if we say that rapidity of work and the production of steel in ever-increasing quantities does not give time enough for the steel to set and prevents slag inclosures and other impurities from rising to the surface, thus preventing maximum cohesion of the crystals? Hence, incomplete crystalline structure and the interposition of a film of impurities, however fine, between the faces of the crystals, hinders, if it does not altogether prevent, the formation of distinct individual breaks in detail fractures, so characteristic in former years.

If the deduction as to the relation between distinct and indistinct detail fractures and change in rapidity and mass-production of steel is correct, and there is no doubt that certain characteristics of detail fractures have disappeared entirely; if experience is worth anything, then we have to modify the conclusion frequently interjected into the rail question, that increased load of cars and locomotives and increased speed of trains is to a large extent responsible for modern rail breakages.

It has been freely remarked all these years that lighter rails of former years did not break so freely as the modern heavier rails. If weight of cars and locomotives and speed of trains are playing such a large factor of destruction now, as seems to be assumed, why did the engineer of former days allow so much larger factor of safety than now? Were the forces of destruction better known and better calculable quantities then than they are now, with all our advance in physics and metallurgy?

If in former years detail fractures covering two-thirds and even more of cross-section of a 3.75-in. axle were not uncommon, and such beautiful fractures have entirely disappeared with the 5.25-in. axles, what is the reason? Were the destructive forces relatively less with the much smaller dimensions than they are with the larger dimensions? Were the shocks and blows and vibrations, producing detail fractures with distinctly marked individual breaks, relatively less severe upon small dimensions than upon modern large dimensions, produc-

ing mushy detail fractures? Is there a relatively greater proportional effect of shocks and vibrations now than there used to be, or has the efficiency of the engineer to calculate the effect of stresses and strains decreased in proportion as loads and speed increased?

Now, then, are we justified in making the extent and characteristics of not one, but hundreds of detail fractures of all classes of structural material, the basis of a criterion of the quality of the material used in these structures? If the artificial tensile test and the very crude and antiquated and unscientific "drop of the beam" are accepted by engineers and mill-men as a safe criterion of the quality of product desired, the unmistakable appearances indicative of the death struggle of the metal against the extraneous forces of destruction, recorded upon the face of the fracture like the print in a book, should be considered even more authentic than any artificial quality measure.

Allowing this reasoning of the value of everlasting comparison to stand, then we are forced to the conclusion that the difference in the character of detail fractures in former days and those of to-day may be found, and has been found, in the greater freedom from slag inclosures and other impurities in the steel of former years and a cleaner, more pronounced crystalline structure of the metal. Rapidity of work seems to interfere with the formation of a clean-cut crystalline structure, such as was found to be an accompaniment of the finest specimen of detail fractures, and mass-production seems to introduce into the steel, and distribute between the faces of the crystals, microscopic films of impurities, which prevent maximum cohesion, producing fissures and seams, with subsequent disastrous results.

The foregoing remarks are offered as facts suggestive of remedies. Whether the obvious remedies can be effectually applied under modern conditions, co-operation between engineers and mill-men will determine. That even a modified application of the remedies would be of great benefit there is no question. To what extent the desired elimination of pipes and honeycombing and sponginess of ingots would contribute their share to increase these benefits, as they undoubtedly would so contribute, experience will tell.

Why Does Lag Increase with the Temperature from which Cooling Starts?

BY HENRY M. HOWE,* NEW YORK, N. Y.

(New York Meeting, February, 1913.)

THE transformation which steel undergoes in slow cooling, from the condition of austenite when above the transformation range into that of pearlite plus either ferrite or cementite below that range, is subject to great lag. This transformation is essentially the pearlitizing of the austenite. Indeed the hardening of steel is due to this lag, which is made so great by extremely rapid cooling that a very large part of the transformation itself is restrained, so that the rapid cooling catches the metal in an only partly transformed state, that of martensite, and under favorable conditions even in an almost wholly untransformed state, as in the cases of Maurer's austenitic manganese steel. Hadfield's austenitic manganese steel, 25 per cent. nickel steel, and other austenitic alloy steels, are austenitic even after slow cooling.

It has long been known that this lag increases (1) with the rapidity of cooling, (2) with the presence of certain retarding elements, notably manganese and nickel, and (3) with the temperature from which cooling starts. The present inquiry touches the last of these influences, but incidentally it throws light on the first also.

This influence of higher heating in increasing lag manifests itself in different ways which, though they may at first seem unrelated, are in fact but different aspects of one and the same thing. Thus, among heatings to different high temperatures, all of them above the transformation range, the higher the temperature to which the steel is heated (1) the lower is the temperature at which the transformation occurs in cooling slowly down, (2) the shorter is the time occupied in the actual cooling from above the transformation range to the room temperature under given external conditions, whether of slow or

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rapid cooling. The effect on the resultant hardness will be considered later. These are manifestations of the increase of lag which the high heating brings about.

There are three effects of high heating which may contribute to this increment of lag. They are, (1) steepening the thermal gradient which exists at the time of cooling through the transformation range; (2) increasing the internal pressure; and (3) what we may call provisionally the effacement of prior crystallization nuclei. In the case of heatings which rise only a moderate distance above a given transformation point there is still a fourth influence which may be recognized, (4) increasing the thoroughness of the diffusion of the proeutectoid element and of the pearlitic ferrite and cementite, so that their re-assembling for the purpose of undergoing the transformation in cooling is the more difficult.

Of these four influences the first two apply to rapid coolings, as in the hardening of steel, the latter two apply to all coolings, whether rapid or slow. Let us take these up in order.

(1) *In the Case of Rapid Cooling, Prior High Heating Steepens the Thermal Gradient at the time of Cooling Through the Transformation Range* and in cooling thence. It is only after the rapid cooling of the outside of the specimen has continued long enough to set up a steep thermal gradient in its interior that the cooling in that interior can become rapid. As, in a series of tests, the quenching temperature is raised progressively higher and higher, the steepness of the thermal gradient within the specimen itself, at the moment of reaching the transformation range, increases correspondingly. Starting with an extremely flat thermal gradient before the quenching begins, at the moment of immersion the skin of the specimen begins cooling very rapidly, and the cooling of the skin continues to be more rapid than that of the axis till that skin has cooled nearly to the temperature of the bath, which is only another way of saying that the thermal gradient is all the while steepening. The important point to note is that this steepening of the thermal gradient may continue even after the cooling of the axis has reached a high speed. Hence even after the quenching temperature has been raised hundreds of degrees above the transformation range, raising it further may still further steepen the thermal gradient. As these successive elevations of the quenching temperature steepen the thermal gra-

dient more and more, so they thereby hasten more and more the cooling of the interior of the specimen in and below the transformation range.

Thus if the quenching starts only at or immediately above Ar_1 , then when the interior is passing Ar_1 it must be cooling relatively slowly, and thus affording some time for the change from austenite into pearlite to begin. Moreover, under these conditions the cooling through the range immediately below Ar_1 , in which the transformation or pearlitizing can still go on, will be appreciably slower than if a steeper thermal gradient has been established by starting the cooling from a higher temperature.

The self-acceleration of the transformation may be considered here before passing on to the second effect of high heating. The starting of the reaction Ar_1 , *i. e.*, the pearlitizing of the austenite, both generates heat and thus retards the cooling, and further by nucleus action hastens itself.

Thus if the quenching starts from immediately above Ar_1 , then at the time when the inside has cooled to just below Ar_1 the existing slowness of cooling caused by the flatness of the thermal gradient just explained gives time for the austenite to begin transforming into pearlite. But this very pearlitizing starts the evolution of heat, and this in turn retards the cooling, thus giving more time for more pearlitizing to go on. Moreover, every particle that has transformed even part way from austenite towards pearlite hastens by nucleus action the pearlitizing of all its neighbors.

It is very important to understand clearly that this pearlitizing is self-propagating, self-accelerating, in this double way. First, it retards the cooling by the heat it evolves, and thus prolongs the sojourn at a temperature high enough for its own continuance. Second, every particle which has pearlitized hastens by nucleus action the pearlitizing of its neighbors. This self-accelerating action is revealed by the very nature of the recalcrescence itself, a reaction which first simply retards the cooling, then arrests it, and then, bursting forth as it were into a veritable flame, evolves heat so rapidly as actually to raise the existing temperature in spite of the fact that the surrounding furnace walls are all the time cooling further, and thus ever exerting a stronger and stronger cooling action on the specimen. The self-acceleration of the pearlitizing enables it to

evolve enough heat to overcome this increasing cooling action of the furnace walls, and to turn a cooling into a rise of temperature so great as to force itself on our attention.

What has been said of the recalescence applies, though with less force, to that part of the transformation which consists in the precipitation of alpha ferrite out of gamma austenite, itself an exothermic reaction as is familiarly known through the retardation A_3 of the cooling curve.

(2) *The Steepening of the Thermal Gradient Caused by High Heating Restrains the Transformation through Increasing Pressure.*—To steepen the thermal gradient means to increase the amount by which, at each instant during the cooling, the temperature of the skin has fallen below that of the axis, and hence the amount by which the contraction due in the skin because of cooling exceeds the contraction due in the axis, and hence the pressure which the excess of contraction of the skin throws on the axial metal.

But this pressure of the outer part of the specimen on its inner part opposes the transformation from austenite into pearlite in that inner part as pointed out by Osmond and Benedicks, because that transformation is accompanied by marked expansion; for it is a general law that the existence of pressure obstructs any transformation which is accompanied by increase of volume, or in other words any transformation which has to overcome that pressure before it can occur.

This pressure should be very marked at the time when its effect in restraining the transformation is greatest, that is when the temperature is at and slightly below the transformation range, a time when transformation would naturally be very rapid. It is of course true that at the very end of the cooling the interior cools faster than the exterior, changing the stress to a great extent. But this seems to be only when the temperature has fallen so low that transformation goes on but slowly.

This effect of raising the quenching temperature, its restraining the heat-evolution due to pearlitizing by increase of pressure, seems to me the most important of the means through which it hastens the passage through the transformation range in rapid cooling. For once the quenching temperature has risen, say 100° above the transformation range, so that by the time when the temperature has fallen to the transformation range the thermal gradient is already steep, it seems that its further

steepening by further raising of the quenching temperature should have a stronger effect through further increase of pressure than through further hastening of the already very rapid cooling, at least in the case of thin or moderately thin specimens, especially in view of the self-accelerating nature of the recalescence.

(3) *The Crystallization Influence of Long and High Heating.*—Brayshaw¹ has shown conclusively that a previous high heating delays the progress of the transformation even during a very slow cooling. This naturally connects itself with the law that high heating restrains the coagulation of the ferrite or cementite precipitated in slow cooling, enunciated by Boynton² and fully confirmed by my observations. Indeed, I find that a great prolongation of exposure even to a temperature as low as 900° has this same effect.

What influence it is that causes this effect is uncertain. Benedicks³ credibly refers it to a progressive wiping out of the centers of crystallization. He noted that, when certain substances which when melted yield at first liquid crystals, were heated barely high enough to turn those crystals into an isotropic, *i.e.*, unoriented mass, and again cooled, those same crystals reappeared with exactly the same orientation and boundaries as before, showing that the nuclei of crystallization persisted even after the crystalline structure seemed to have disappeared entirely. But if the heating rose much higher, then on cooling again the new crystals which now formed bore no relation to the former ones, showing that the higher heating had destroyed the old nuclei of crystallization. He infers a like action in the heating of the austenite to temperatures farther and farther above the transformation range, and he refers the lag of transformation thus caused to the lessened number of nuclei whence transformation can proceed, for the rapidity of transformation, as Tamman has pointed out, increases not only with the temperature gap below the true transformation temperature but also with the number of nuclei of transformation existing.

I had referred this effect of high heating to its increasing the perfection and hence the stability of the crystalline organi-

¹ *Proc. Inst. Mech. Eng.*, 1910, 2, pp. 540-541.

² *Iron and Steel Magazine*, 7, 1904, p. 472.

³ *Proc. Inst. Mech. Eng.*, 1910, 2, p. 703.

zation,⁴ a stability which opposed the transformation and hence induced lag.

The wiping out of the nuclei previously existing, as postulated by Benedicks, and the perfecting of the crystalline organization of the austenite which I invoked, seem really to be two aspects of the same thing; for to remove the older crystalline influence and thus leave the crystalline influence of the austenite unhampered is in effect to strengthen it. But of the two Benedicks's point of view is the more instructive because the more precise.

(4) *High Heating Increases the Diffusion of the Several Constituents.*—The higher the temperature rises above A_1 the farther does diffusion undo the initial localizing of the carbon in the sheets of pearlitic cementite, and hence the less readily does the cementite thus scattered re-assemble into the form of crystals of pearlitic cementite on again cooling past A_1 .

A like effect follows as regards the pro-eutectoid ferrite. When the temperature first rises above A_3 the very spots which had been occupied by pro-eutectoid ferrite are still poorer in carbon than the rest of the austenite. They are in fact equivalent to the austenite of a steel poorer in carbon than that normal to the steel on which we are operating. But it is a most familiar fact that the lower the carbon content the more readily does the transformation occur. For instance, a rapid cooling which hardens a eutectoid steel glass hard causes only very little hardening in a steel of 0.05 per cent. of carbon. In that high heating effaces the more completely these local impoverishments in carbon, which facilitate transformation, it in effect obstructs transformation, or in other words increases lag.

This effect of high heating may be important when a brief heating to immediately above the transformation range is replaced by one to a temperature say 50° or even 100° higher. But this diffusion should soon complete itself so fully that still higher heating should have no important effect in this respect.

A Higher Quenching Temperature should Increase Hardness through Increasing Stress.—In the case of rapid coolings still a fifth influence of raising the quenching temperature contributes to increase the resultant hardness, viz., the residual stress. This

⁴ *Proc. Amer. Soc. for Testing Materials*, XI, p. 353, 1911.

will naturally be the more severe the higher the quenching temperature; and, though this stress is clearly not the chief immediate cause of hardening, as is shown by the fact that, under like conditions of quenching, the hardness is the greater the thinner the specimen quenched, yet the mere existence of stress should contribute to the hardness, and any increase of stress should increase this contribution.

Effect of Raising the Quenching Temperature on the Resultant Hardness.—Though raising the quenching temperature A_1 upwards has been shown clearly by Benedicks⁵ to increase the rapidity of cooling from 700° down, and thus to restrain the transformation, and though the fact that austenite is best had by quenching from a very high temperature further confirms this, we are not to infer that the resultant hardness of the quenched steel need increase as the quenching temperature is raised, after this raising has passed materially above the transformation range. For the beginning of the transformation, from austenite to martensite, is accompanied by an increase in hardness, to be followed by a great loss of hardness as the transformation proceeds farther to the stages of troostite and sorbite. Moreover, the coarsening of the structure which higher heating causes may lessen the hardness, and so may surface decarburization if the skin only of the specimen is tested.

Indeed, the very cause which throws the interior into compression and thereby restrains the transformation there must simultaneously throw the skin into tension and thereby favor the transformation there, an inference which is supported by Benedicks's observations.⁶ Though the skin cools the most rapidly yet the transformation there goes farther than in the interior, for in the case of a carbon steel he finds untransformed austenite only in the interior, the very skin always transforming as far as martensite even in the most energetic quenchings.

Hence we are not to be surprised that Brinell,⁷ Shore,⁸ and Brayshaw⁹ agree in reporting a decrease of hardness as the quenching temperature rises beyond the transformation range.

⁵ *Jour. Iron and Steel Inst.*, 1908, ii, p. 218.

⁶ *Journal Iron and Steel Inst.*, 1908, ii, p. 244.

⁷ *Jour. Iron and Steel Inst.*, 1901, i, p. 273.

⁸ *The Shore Scleroscope, Its Application to the Arts*, 101, 1910, p. 44.

⁹ *Proc. Inst. Mech. Eng.*, 10, 1910, No. 2, p. 561.

Comparative Notes on Steel-Rail Rolling.

BY ROBERT W. HUNT, CHICAGO, ILL.

(New York Meeting, February, 1913)

I HAVE frequently stated that while the chemical composition of steel is important, yet even greater importance is connected with the mechanical and heat treatment of the metal. During the past year I encountered such a positive example of that fact that I deem it worth putting upon record.

A prominent railway system divided an order for open-hearth steel rails between two steel works, both of which are under the control of the same corporation; giving to one about 18,000 tons, and to the other 7,500 tons of the same section and to be made under the same specifications. The rolling results obtained in the two mills varied so widely that a study of the figures is intensely interesting and serves to illustrate the advantages obtained by careful ingot casting, and, quite as pertinently, the possible benefits of careful subsequent heating and rolling with moderate reductions in the rolling process. The smaller order was rolled complete in four installments during the same months that the larger order was being made in six separate installments; but, to permit of exact comparisons, the totals of the four installments (completing the order with the mill which I will call A) are given with the totals of the first four installments of the larger order, made by the mill designated as B. These rolling results are:

	Mill A.	Mill B.
Number of rollings,.....	4	4
Total number of rails rolled.....	18,278	27,832
Percentage of rails cut to short lengths because of flaws near ends, etc.....	0.7	7.6
Percentage of rails made second quality for flaws, etc.	0.9	6.3
Percentage of rails scrapped for flaws, etc.....	0.6	4.5
Percentage of rails scrapped for failure at drop test...	none	3.4

It should be stated that the figures given above cover all the reasons for putting the rails in the classes stated. Thus a part of the percentage of rails cut to short lengths may have been

because of bad drilling or bad sawing. Such classification, however, in this case is entirely proper, as neither mill suffered from unusual or abnormal difficulties in any way, and the figures indicate ordinary performance, uninfluenced by unusual errors of workmanship or mechanical troubles in rolling.

Emphasis should be laid on the fact that both mills were working to exactly the same specification, and producing a section which has been in use for several years and in large tonnage. It is an 85-lb. section having 36.7 per cent. of metal in the head, 22.2 per cent. in the web, and 41.1 per cent. in the base, being, therefore, well proportioned for the avoiding of torn flanges or other rolling difficulties, sometimes encountered with sections having thin flanges.

The chemical composition and drop test specified, with the average results obtained, were:

	<i>Composition.</i>		<i>Average Obtained.</i>	
	<i>Specified.</i>		<i>Mill A.</i>	<i>Mill B.</i>
	<i>Per Cent.</i>		<i>Per Cent.</i>	<i>Per Cent.</i>
Carbon.....	0.55 to 0.70		0.62	0.61
Phosphorus, not over.....	0.045		0.023	0.023
Manganese.....	0.65 to 0.95		0.83	0.74
Silicon.....	0.075 to 0.18		0.13	0.14
Sulphur, not over.....	0.055		0.029	0.039
<i>Drop Test.</i>				
Deflection specified.....	min., 1.75 in.; max., 3.25 in.			
	<i>Results Obtained.</i>			
	<i>Mill A.</i>	<i>Mill B.</i>	<i>Min.</i>	<i>Max.</i>
Deflection on 4-ft. supports with 2,000-lb. tup falling 20 ft., inches.....	2.3	3.2	2.2	3.8
Average deflection, inches.....	2.7	2.8
Number test pieces breaking on first blow.....	0	9
Number test pieces showing pipe or segregation when nicked and broken.....	3	9
Number heats rejected because of two pieces failing at drop test.....	0	2
Number heats rejected for exceeding deflection limits at drop test.....	0	4
Number heats rolled.....	119	173

Both mills took advantage of the full range permitted in the chemical limits, and how successful they were in obtaining consistent carbon results is shown by the table below.

	0.55	0.56	0.57	0.58	0.59	0.60	0.61	0.62	Total.
Permitted carbon, p. ct.									
No. of heats : mill A... ..	1	2	4	5	17	17	14		
No. of heats : mill B... ..	20	8	9	14	12	16	14	14	
Permitted carbon, p. ct.	0.63	0.64	0.65	0.66	0.67	0.68	0.69	0.70	
No. of heats : mill A... ..	13	14	11	8	9	4	119
No. of heats : mill B... ..	14	15	7	7	10	6	6	1	173

It will be noted that the results obtained by mill A were more consistent than those of mill B.

The chemical compositions obtained at the two mills agree so closely that some other reason must be sought to explain the divergent physical results obtained in the product, and, therefore, data on the actual performance of the mill operations is important. The principal items are :

	Mill A.	Mill B.
Number of furnaces making the steel.....	6	27
Average size heats cast.....	196,000 lb.	183,500 lb.
Maximum weight heats cast.....	214,000 lb.	220,000 lb.
Minimum weight heats cast.....	128,000 lb.	155,000 lb.
Time metal held in ladle before casting :		
Average.....	4 min.	3.5 min.
Maximum.....	11 min.	52 min.
Minimum.....	1 min.	2 min.
Number ingots per heat :		
Average.....	20	23
Maximum.....	22	28
Minimum.....	13	20
Time between casting and stripping :		
Average.....	43 min.	1 hr. 7 min.
Maximum.....	2 hr. 15 min.	5 hr. 55 min.
Minimum.....	24 min.	15 min.
Time between stripping and charging into soaking pits :		
Average.....	16 min.	46 min.
Maximum.....	1 hr. 6 min.	11 hr.
Minimum.....	3 min.	10 min.
Time between charging in soaking pits and blooming :		
Average.....	2 hr. 20 min.	2 hr. 47 min.
Maximum.....	4 hr. 30 min. ^a	6 hr. 5 min. ^b
Minimum.....	1 hr. 30 min.	1 hr. 0 min.
Number cold heats charged.....	8	None
Per cent. top discard.....	10 to 13 p. c.	10 to 12 p. c.
Time in reheating furnaces.....	20 to 50 min.	None ^c
Average weight ingots.....	9,740 lb.	7,970 lb.
Size of ingots.....	22 by 26 in.	20 by 24 in.
Number passes in blooming mill.....	21 to 25	9
Size of bloom produced.....	7 by 9 in.	8 by 8 in.
Number passes in rail mill.....	11	9
Total passes.....	32 to 36	18

^a Indicates one heat in pits 25 hr.

^b Indicates one heat in pits 24 hr.

^c Mill B rolls direct, does not reheat.

It will be especially noticed that both mills were casting large heats, approximating 85 tons. These were made by almost identical methods of the usual scrap and pig iron process, the iron taken from mixers, as required, and varying in proportion to the scrap used. As far as possible, at both mills, this mixer metal was used for recarbonizing in the furnace, but many heats had coke or coal added to the ladle on tapping, in addition to the usual ferro-manganese and ferro-silicon. While the heats at mill A were slightly larger than at B, the difference is not of importance, but it is pertinent to note that there are but 6 furnaces represented at mill A as against 27 which furnished the rail steel at B; and consequently there were probably but two steel melters working at A as against at least six at B. The personal equation may, therefore, have played an important part in making the steel.

Equally noticeable is the fact that while the average for the time intervals of the various operations at the two mills is not much different, still the range between the maximum and minimum for B is consistently greater than for A. There can be but little doubt but that a delay between casting and stripping ingots and between stripping and charging in the soaking pits is likely to be seriously reflected in the soundness of the ingots, and of at least equal importance is the necessity for teeming the steel at uniform temperature. A variation of 50 min. in the time heats were held in the ladle prior to casting must have caused variable teeming temperatures, and produced many blow-holes in some of the ingots. Admitting the presence of blow-holes near the surface of the sides of the ingots, and remembering the oxidizing action in the soaking pits, it is not surprising that heavy reductions in the blooming mill had an extremely detrimental effect on the product. This is reflected, no doubt, in the large number of rails containing flaws found at mill B.

Appreciating, therefore, the probable difference in the ingots produced at the two mills and the actual difference in the blooming practice, the principal other variable existing was in the reheating of the blooms at mill A. There the ingots were cut into four blooms, which were then given a wash heat in reheating furnaces, after which they were rolled into two rails

in a rail train of 11 passes. At mill B the rail train consisted of 9 passes, with no reheating of the blooms.

It is interesting to note the reported history of the heats rejected at mill B. Two heats were rejected at the drop test because two of the three test pieces broke on the first blow of the tup. The steel of the first heat was reported as being somewhat low in temperature when tapped. While pouring, the nozzle froze up; and while the ingots were in the soaking pits an average of 4 hr. and 20 min., they bloomed cold and rough. The second heat was quite the opposite, the tapping temperature being high, and while casting the stopper head was finally lost; but the heating and the blooming were normal.

Four heats were lost at the drop test because they exceeded the deflection limits. On the first, all conditions were reported normal, save that the ingots were held in the pits for an average time of 24 hr. The second heat tapped cold, the pouring nozzle froze, and the ingot tops were spongy. The third tapped hot, but otherwise, and like the fourth, had normal conditions.

Analysis of rejected heats gave:

C.	P.	Mn.	Si	S	Remarks.
0.70	0.015	0.74	0.11	0.034	Two pieces broke.
0.57	0.020	0.65	0.08	0.035	Two pieces broke.
0.55	0.014	0.65	0.08	0.036	Deflection over limit.
0.55	0.017	0.65	0.09	0.037	Deflection over limit.
0.60	0.026	0.71	0.11	0.041	Deflection over limit.
0.57	0.026	0.75	0.08	0.051	Deflection over limit.

DISCUSSIONS.

Discussions of papers contained in this volume have been placed immediately after the respective papers with the exception of the one here printed, which was received too late to be incorporated with the paper to which it relates.

The Sulphide Ores of Copper. Some Results of Microscopic Study.

Discussion of the paper of L. C. Graton and Joseph Murdoch, p. 26.

THOMAS T. READ, New York, N. Y. (communication to the Secretary *):—At the meetings of English technical societies it not infrequently happens that, during the discussion of a paper, someone will arise to say that although he highly appreciated the material presented, nevertheless he believed it would have been better if some other phase of the question had been more highly emphasized. Assuming that this form of discussion is permissible in the Institute, I should like to reiterate the point made by Prof. A. J. Moses that conclusive proof of the identity of the minerals in question is an essential feature of the study of ores in this way. This is also expressed by Paul Krusch in his recent valuable paper¹ on the sulphide ores (in which he arrives at conclusions diametrically opposed to those of Mr. Graton), as follows: "The diagnostic properties of the various ore minerals are known to but a limited circle of investigators." Many earlier investigators began the study of ores beneath the microscope by reflected light, but discontinued their work, in most cases, because they were unwilling to undertake the great amount of preliminary work necessary to securely establish a basis for their inferences. It is evident that Messrs. Graton and Murdoch realize that the promised later paper on diagnostic characters will be the more important of the two, but the point should be emphasized to forestall a flood of literature from other investigators who may gain the impression that the brief survey of polished specimens is a *vade mecum* to incontrovertible conclusions as to any given ore deposit.

* Received Sept. 25, 1913.

¹ *Mining and Scientific Press*, vol. cvii, No. 11, pp. 418 to 423 (Sept. 13, 1913).

On p. 35 of this paper the statement is made that the great volume of laboratory study has prevented a corresponding study of the literature of the subject. This doubtless serves to explain the footnote on p. 65, where only the recent paper by A. C. Spencer is cited as corroborating the view that secondary enrichment is essentially a process of oxidation. In the *Transactions* for 1906² I stated the following conclusion from a chemical study of enrichment. "It is clear, then, that oxidation alone is sufficient to produce the enriched sulphides occurring in the zone of secondary enrichment." The fact that the chemical work on which this conclusion was based was somewhat faulty does not detract from the soundness of the conclusion, and it is the more gratifying to find it substantiated by other methods of work. While believing that oxidation has been more important than reduction in the formation of the rich secondary sulphides, it is probable that the latter is at times important. The distinction Krusch and others make between oxidation and cementation ores may at times be of service, though in others it is unquestionably a source of added confusion, unless the term cementation is strictly defined. The appearance of two such important papers in quick succession is pleasing evidence that the study of these important and puzzling ores is being followed with unabated vigor.

² *Trans.*, xxxvii, 302 (1906).

INDEX.

[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

- Adams, Frank D.: flow of rocks, 19.
- Alabama: mineral production (1911), 144.
- Alabandite: effect in precipitating silver from sulphate and gold from chloride solutions, 227.
- Alaska: mineral production (1911), 144.
- Aluminum, silicon and ferro-titanium as oxidizers of steel, 401, 428, 436, 497.
- American Eagle gold-mine, Elk City district, Idaho, [120].
- American Institute of Mining Engineers:
- activities for 1913, xiii.
 - committees, ix.
 - honorary members, viii.
 - meetings: New York, February, 1913, xv.
 - membership, xiv.
 - officers, vii.
- Analyses:
- blast-furnace slag, 329.
 - fire-clays, 127, 129, 130, 134.
 - steel: deoxidized with thermit, 460.
 - open-hearth, 438-440.
 - steel axle: open-hearth, 441.
 - steel rail from compressed ingot, 404.
 - water from Vulcan iron-mine, Michigan, 9.
- Argall, Philip: ball mills for crushing, 204.
- Arizona: mineral production (1911), 144.
- Arkansas: mineral production (1911), 144.
- Arsenides:
- cobalt and nickel: association with native silver, 235.
 - effect in precipitating silver from sulphate solutions, 227.
- Arsenopyrite: effect in precipitating silver from sulphate and gold from chloride solutions, 227.
- Assaying:
- mine waters, 3.
 - silver: Gay-Lussac method, 256-271.
- Assays: silver bullion, 262-271.
- Austin, L. S.: return on mining investments, 312.
- Ball mills: conical vs. cylindrical, 204.
- BASTIN, EDSON S.: *Discussion on The Sulphide Ores of Copper*, 86-87.
- BASTIN, EDSON S., and PALMER, CHASE: *The Role of Certain Metallic Minerals in Precipitating Silver and Gold*, xvi, 224-238.
- Bayley, W. S.: contact phenomena of rocks, 14.

- BECK, E. A.: *The Use of Anti-Piping Thermit in Casting Steel Ingots*, xviii, 453-460.
- Beck, R.: association of native silver and copper sulphide ores, [234].
microscopic examination of minerals, [28].
- Benedicks. influence of heating on crystallization of steel, [520].
- Bessemer medal: award to Floris Osmond, [274].
- Bessemer steel: see Steel.
- Bibliographies:
ferrite formation, 222.
scientific writings of Floris Osmond, 278.
- Biographical Notice of Floris Osmond* (SAUVEUR), xix, 272-281.
- BIRKINBINE, J. L. W.: *Discussion on Valuation of Iron-Mines* 299.
- BIRKINBINE, JOHN: *Discussions: on Notes on Cast-Iron*, 376.
on Valuation of Iron-Mines, 297-300, 303.
- Blast-Furnace Slag-Analyses for 24 Hours* (GRAMMER), xix, 327-329.
- Blast-furnace practice: record of a day's run on basic and Bessemer iron, 329.
- Blende: formation of ferrites in roasting, 210-223.
- Blende and marcasite: Friedrich's freezing curve, 213
- Blow-holes in steel: see Steel.
- Boiler-plate steel: formation of blow-holes, [424].
- Boria mica-mine, Hazaribagh district, India, [98].
- Bornite:
effect in precipitating silver from sulphate and gold from chloride solutions. 227.
microstructure, 45.
relation with chalcocite in copper ores, [53].
- Boynton: influence of heating on crystallization in steel, [520].
- Brayshaw: effect of temperature of quenching on hardness of steel, [522].
influence of heating on crystallization in steel, [520].
- Brinell: effect of temperature of quenching on hardness of steel, [522].
- Brinell hardness tests of rail from Talbot process steel, 422.
- BRINSMADE, R. B.: *Discussion on Valuation of Iron-Mines*, 322-326.
- British Columbia: fire-clay deposits. 135.
- BROOKS, G. S.: *Notes on the Formation of Ferrites in Roasting Blende*, xvii, 210-223.
- Brun, Paul: gases in rocks, 15.
- Buehler, H. A., and Gottschalk, V. H.: oxidation of sulphides, [225].
- Bunker Hill and Sullivan mine, Idaho: increase of mine water with depth, 22
- BURNS, E. Z., and SIMONDS, F. M.: *A Problem in Mining, together with Some Data on Tunnel-Driving*, xvi, 147-180, 184-188.
- Buster gold mine, Elk City district, Idaho, [116].
- California: mineral production (1911), 144.
- Campbell, William: microscopic examination of minerals, [28].
- Campbell, William, and Knight, C. W.: microstructure of pyrrhotites, [41].
- Canada: fire-clay deposits, 123-136.
- Canaris, C.: tests of anti-piping thermit, 454.
- Carbon: segregation in steel ingots, 438.
- Cast-iron: see Iron.
- Cement: production of U. S. by States (1911), 144.
- Cementite: behavior in steel, 375.
- Chalcocite:
association with native gold in Leonard mine, Butte, Mont., 236.
effect in precipitating silver from sulphate and gold from chloride solutions, 227.
microstructure, 52.
reactions with silver sulphate solutions, 231.
relation with bornite in copper ores, [53].
test for identification, 232.

- Chalcopyrite: effect in precipitating silver from sulphate and gold from chloride solutions, 227.
 microstructure, 41.
- Chamberlain, R. T.: gases in rocks, 15.
- Chilean mills vs. Hardinge: competitive tests, 201.
- Cinder and cinder agglomerates microstructure, 330-345.
- CLAMER, G. H.: *Discussion on Sound Steel Ingots*, 494-495.
- Clay deposits. Canada, 123-136.
- Clay products: production of U. S. by States (1911), 144.
- Coal:
 cost of production, and profit, U. S. (1909), 138.
 production of U. S. by States (1911), 144.
- Coal-lands. taxation of, 300.
- Cobalt and nickel arsenides: association with native silver, 235.
- Cobaltite: effect in precipitating silver from sulphate and gold from chloride solutions, 227.
- Colles, George W.: mica and the mica industry, [95].
- Colorado:
 mineral production (1911), 144.
 Rawley silver-copper mine, Saguache county, 147-188.
 Up-to-Date copper mine, Caribou, Boulder county, 234.
- Commercial Production of Sound Steel Ingots* (GATHMANN), xviii, 461-472.
- Committees of the Institute, ix
- Comparative Notes on Steel Rail Rolling* (HUNT), xix, 523-527.
- Comstock lode: ground-water level, 7.
- Conical mills. Hardinge, 194-209.
- Connate waters, 8.
- Copper:
 influence in precipitation of silver from solutions, 228.
 production of U. S. by States (1911), 144.
- Copper-bearing minerals: list of, 34.
- Copper minerals:
 effect in secondary enrichment of gold and silver deposits, 232.
 order of formation, 65.
- Copper mines:
 Gagnon, Butte, Mont., 233.
 Leonard, Butte, Mont., [236].
 Up-to-Date, Caribou, Colo., 234.
- Copper ores:
 association of sulphide with native silver, 232.
 microscopic study, 26-33.
 primary and secondary minerals, 63.
- Costs:
 coal production, U. S. (1909), 138.
 producing sound steel ingots by Hadfield and other methods, 477.
 steam, gas, and electric power, Sonora, Mexico, 190.
 tunneling operations, Colorado, 172-179.
- Cox, A. J.: solubility of silver salts, [227].
- Covellite:
 effect in precipitating silver from sulphate and gold from chloride solutions, 227, 231.
 microstructure, 50.
- Crucible steel: see Steel.
- CUSHING W. C.: *Discussions: on The Production of Solid Steel Ingots*, 409-410.
on Sound Steel Ingots, 491-494.

- CUSHMAN, ALLERTON S.: *Discussion on Sound Steel Ingots*, 495-497.
- Daly, R. A.: igneous intrusions, 13.
- DARTON, N. H.: *Discussion on The Ground-Waters*, 24.
- Denny, G. A.: return on mining investments, 311-315.
- Deoxidizers for steel:
- aluminum, titanium, and silicon, 401, 428, 436, 497.
 - thermit, 453-460.
- de Schmid, Hugo S.: mica occurrence and uses, [95].
- DEWEY, FREDERIO P.: *The Gay-Lussac Method of Silver Determination*, xvii, 256-271.
- DIXON, A. FAISON: *The India Mica Industry*, xvii, 94-112.
- Dormacola mica-mine, Hazaribagh district, India, [99].
- DUDLEY, P. H.: *Piping and Segregation of Ingots of Steel and Ductility-Tests for Open-Hearth Steel Rails*, xviii, 424-452.
- Discussions: on The Production of Solid Steel Ingots*, 408-409.
- on Sound Steel Ingots*, 506.
- Dugi mica-mine, Hazaribagh district, India, [98].
- Dwight and Lloyd sinter: microstructure, 341.
- Earthquakes and mountain formation: See's theory, 12.
- Effect of High Carbon on the Quality of Charcoal-Iron* [*Trans.*, xlv, 314]. *Discussion* (JOHNSON), 391.
- Electric furnace: Hering: method of operating, 494.
- Electric Power Installation at El Tigre, Sonora, Mexico* (MALCOLMSON), xvi, 189-193.
- Electricity: oxidation of sulphides by, 225
- Elk City Mining-District, Idaho County, Idaho* (FLAGG), xvii, 113-122.
- Emmons, S. F.: association of native silver with cobalt and nickel arsenides, 235.
- Emmons, W. H.: metamorphism of pyritic copper ores, [31].
- Enargite:
- effect in precipitating silver from sulphate and gold from chloride solutions, 227.
 - microstructure, 59.
- Ferrites:
- bibliography, 222.
 - formation in roasting blende, 210-223.
- Ferro-titanium, aluminum, and silicon as deoxidizers of steel, 401, 428, 436, 497.
- FINLAY, JAMES R.: *Valuation of Iron-Mines*, xix, 282-297, 321-322.
- Finlayson, A. M.: pyritic deposits of Huelva, Spain, [236].
- Fire-box steel: formation of blow-holes, [424].
- Fire-clay: analyses, 127, 129, 130, 134.
- Fire-Clay Deposits of Canada* (RIES), xvii, 123-136.
- FLAGG, ARTHUR L.: *The Elk City Mining-District, Idaho County, Idaho*, xvii, 113-122.
- Florida: mineral production (1911), 144.
- Flue-dust nodules: microstructure, 340.
- Flue-dust sinter: microstructure, 332, 341.
- FOWLER, GEORGE L.: *Discussion on The Production of Solid Steel Ingots*, 411.
- Friedrich, K.: freezing curve of blende and marcasite, [213].
- Fuller, M. L.: water in the earth's crust, 23.
- Furnace cinder: microstructure, 331.
- Furnaces:
- Hering electric: method of operating, 494.
 - open-hearth, using producer-gas, 346-354.
- Gagnon copper mine, Butte, Mont.: ratio of metals in ore, 233.
- Galena: effect in precipitating gold from chloride solution, 232.
- Gas-operated steel-furnaces, 346-354.

Gases:

- in rocks, 15.
- natural: production of U. S. by States (1911), 144.
- producer vs. natural, for open-hearth furnaces, 352.
- volcanic: water condensed from, 18.

GATHMANN, EMIL: *Commercial Production of Sound Steel Ingots*, xviii, 461-472.

Gay-Lussac Method of Silver Determination (DEWEY), xvii, 256-271.

Geographical Distribution of Mining Development in the United States (PARKER), xvi, 137-145; *Discussion* (KEMP), 146; PARKER), 146.

Geologic work: relations of scale, 26.

Geology:

- Elk City gold-mining district, Idaho county, Idaho, 117.
- Hazaribagh mica-mining district, India, 96.
- London mountain, Park county, Colo., 240.

Georgia: mineral production (1911), 144.

Gold:

- association of native, with sulphide ores, 236.
- precipitating by metallic minerals, 224-238.
- production of U. S. by States (1911), 144.

Gold mines:

Colorado: London, Park county, 239-250.

Idaho: Elk City district:

- American Eagle, [120].
- Buster, [116].
- Mineral Zone, [121].
- Revenue (California), 119.
- South Fork, [120].

Goldschmidt, V. N.: investigations of contact phenomena at Christiania, Norway, 17.

Gottschalk, V. H. and Buehler, H. A.: oxidation of sulphides, [225].

GRAMMER, F. L.: *Blast-Furnace Slag-Analyses for 24 Hours*, xix, 327-329.

GRATON, L. C., and MURDOCH, JOSEPH: *The Sulphide Ores of Copper. Some Results of Microscopic Study*, xvii, 26-81, 88-92.

Gravel: production of U. S. by States (1911), 145.

GRIFFIN, P. H.: *Discussion on Notes on Cast-Iron*, 383-386.

Grinding mills

- Hardinge, 194-209.
- speed of rotation, 201.

Ground-water level, 5, 18.

Ground-Waters (KEMP), xvi, 3-24; *Discussion* (DARTON), 24.

HADFIELD, ROBERT A.: *Plant for Hadfield Method of Producing Sound Steel Ingots*, xviii, 473-478.

Hammond, John Hays: return on mining investments, 313.

HARDINGE, H. W.: *The Hardinge Conical Mill*, xvii, 194-209.

Hazaribagh mica-mining district, India, 95.

Heating-furnace cinder: microstructure, 331.

Heberlein flue-dust agglomerate: microstructure, 332.

Heberlein pyrites cinder agglomerate: microstructure, 339.

Hematite ore nodules: microstructure, 340.

Hering electric furnace: method of operation, 494.

HESS, HENRY: *Discussion on Sound Steel Ingots*, 507-508.

Heumann: action of metallic sulphides in silver nitrate solutions, 230.

HIBBARD, HENRY D.: *Discussions: on New Design of Open-Hearth Steel-Furnace for Producer-Gas*, 351.

on The Production of Solid Steel Ingots, 406, 418-419.

Holland, Sir Thomas: mica-deposits, India, [95].

- Hommel, W.: formation of ferrites in roasting iron and zinc oxides, [214].
 Honorary members of the Institute, viii.
 Horn-Silver mine, Utah: ground-water level, [6].
 HOWARD, JAMES E.: *Discussion on Sound Steel Ingots*, 490-491.
 HOWARD, L. E.: *Discussion on Sound Steel Ingots*, 486-487.
 HOWE, HENRY M.: *Why Does Lag Increase with the Temperature from which Cooling Starts?* xix, 516-522.
 Discussions: on Notes on Cast-Iron, 376-380.
 on The Production of Solid Steel Ingots, 406, 413-418.
 on Sound Steel Ingots, 502-504.
 HUNT, ROBERT W.: *Comparative Notes on Steel Rail Rolling*, xix, 523-527.
- Idaho:
 Elk City mining-district, 113-122.
 mineral production (1911), 144.
 Illinois: mineral production (1911), 144.
India Mica Industry (DIXON), xvii, 91-112.
 Indiana: mineral production (1911), 144.
 Ingot molds: Gathmann, 470.
 Ingot stripper: Gathmann, 467.
 Inikuvti mica-mine, Nellore district, India, [109].
 Institute: see American Institute of Mining Engineers.
 Iowa: mineral production (1911), 144.
- Iron:
 cast: carbon, combined: effect on physical properties, 356.
 carbon, graphitic: effect of shape and mode of distribution, 357.
 effect of rate of cooling on shape of particles, 360.
 effect of silicon on shape of particles, 361.
 composition: proximate, 363.
 structural: 364.
 graphitizing, 373.
 physical properties vs. ultimate composition, 355.
 physical tests, 380.
 solidification, 368.
 temperature range of solidification, 397.
 white spots in gray iron, 366.
 charcoal: effect of high carbon on quality, 391.
 Iron-bearing materials: microstructure of sintered, 330-345.
 Iron-mines: valuation, 282-326.
 Iron-ore:
 Brazilian: cost factors, 289.
 Cuban: cost factors, 288, 307.
 future prices, 292, 308, 319, 325.
 influence of taxes on cost, 299.
 Lake Superior: standard grades, 288.
 cost factors, 288, 307, 320.
 prices (1890-1912), 292, 309.
 production of U. S. by States (1911), 144.
 sintered: microstructure, 340.
 Itakara mica-mine, Nellore district, India, 109.
- JOB, ROBERT: *Discussion on Sound Steel Ingots*, 482-484.
 JOHNSON, J. E., JR.: *Discussions: on Notes on Cast-Iron*, 386-390.
 on The Effect of High Carbon on the Quality of Charcoal-Iron [*Trans.*, xlii, 314], 391.

Kansas: mineral production (1911), 144.

KEMP, JAMES F.: *The Ground-Waters*, xvi, 3-24.

Discussions: on The Geographical Distribution of Mining Development in the United States, 146.

on A Problem in Mining, together with Some Data on Tunnel-Driving, 184.

on The Sulphide Ores of Copper, 82-84, 92.

secondary enrichment of ores, [65].

KENNEY, E. F.: *Discussion on The Production of Solid Steel Ingots*, 407-408.

Kentucky: mineral production (1911), 144.

Kick's law: power for dividing particles of matter, 196.

KING, L. V.: strength of rocks under stress, 19.

KIRCHHOFF, CHARLES: *Discussion on The Production of Solid Steel Ingots*, 413

KLUGH, B. G.: *Microstructure of Sintered Iron-Bearing Materials*, xix, 330-345.

KREUZPOINTNER, PAUL: *Discussion on Sound Steel Ingots*, 513-515

Laboratory-work:

Gay-Lussac method of silver determination, 256-271.

sampling ore containing coarse gold, 251-255.

Lane, A. C.: field assay of mine waters, 3.

Laney, F. B.: relation of bornite and chalcocite in copper ores, [53].

Lavoisier medal: award to Floris Osmond, [273].

Lazarevic: color change of covellite, [62].

Lead: production of U. S. by States (1911), 145.

Leonard copper mine, Butte, Mont.: native gold associated with chalcocite, 236.

Lincoln, F. C.: magmatic emanations, 15

microstructure of chalcopyrite, [44].

Lindgren, W.: nature of replacement, [67].

report on Tertiary gravels of California, [238].

Lime: production of U. S. by States (1911), 145.

LOCKE, CHARLES E.: *School Laboratory-Work: Sampling of an Ore Containing Coarse Gold*, xvii, 251-255.

London Mine, Mosquito Mining-District, Park County, Colo. (MOORE), xvii, 239-250.

Louisiana: mineral production (1911), 144.

Lowe's solution, 212.

Lucky Tiger Combination Gold Mining Co., Kansas City, Mo.: electric power installation, El Tigre, Sonora, Mexico, 189-193.

Lugunwa mica-mine, Hazaribagh district, India, [98].

Magmas:

intrusive processes, 12.

stopping and absorption by, 12.

Magmatic waters, 10.

condensed from volcanic gases, 18.

investigations of Brun, 15.

MALCOLMSON, JAMES W.: *Electric Power Installation at El Tigre, Sonora, Mexico*, xvi, 189-193.

Manganese: segregation in steel ingots, 438.

Maps:

Canada: fire-clay deposits, 124.

Colorado: Rawley silver-copper mine, 148.

Idaho: Idaho county, 114.

Marcasite: effect in precipitating silver from sulphate and gold from chloride solutions, 227.

Marcasite and blende: freezing curve, [213].

Marmatite: formation of ferrites in roasting, 210-223.

- Maryland: mineral production (1911), 144.
 Massachusetts: mineral production (1911), 144.
 Mattes: zinc-iron: formation of ferrites in roasting, 210-223
 Mesabi iron range, 286.
 Meteoric waters, 4.
 Meunier, S.: action of gold and silver solutions on metallic sulphides, [230].
 Membership of the Institute, xiv.
 Mica.
 imports, U. S. (1909), and value, 112.
 market and prices, London, 111.
 mining, 94-112.
 mining laws, India, 107.
 preparing for market, 106, 110.
 production (1910), India, 112.
 Mica-mines:
 India: Hazaribagh district:
 Boria, 98,
 Dormacola, 99
 Dugi, 98.
 Lugunwa, 98.
 Pania, 98
 Phutlai Bogda, 98.
 Torna, 98.
 Nellore district:
 Inikuvti, 109.
 Itakara, 109.
 Sankawa, 110.
 Michigan: mineral production (1911), 144.
 Microscopic examination of ores:
 preparation of specimens, 32.
 study of copper ores, 26-93.
Microstructure of Sintered Iron-Bearing Materials (KLUGH), xix, 330-345.
 MILLER, HERBERT F., JR.: *New Design of Open-Hearth Steel-Furnace Using Producer-Gas*, xix, 346-354
 Discussion on Sound Steel Ingots, 508-509.
 Millerite: Effect in precipitating gold from chloride solution, 232.
 Mine drainage tunnels:
 Arizona Copper Co., Morenci, Ariz.: progress, 182.
 Rawley, Saguache county, Colo., 147-188.
 Mine valuation, 232-326.
 Mine water: variation in quantity with depth, 20
 Mineral production of U. S. by States (1911), 137, 144.
 Mineral Zone gold-mine, Elk City district, Idaho, [121].
 Mines: see name of mine, mining company, or product.
 Mining development in the U. S.: geographical distribution, 137-146.
 Mining investments: return on, 295, 310, 322.
 Minnesota: mineral production (1911), 144.
 Missouri: mineral production (1911), 144.
 MOLDENKE, RICHARD: *Discussion on Notes on Cast-Iron*, 380-383
 Money: present value of future payments, 294, 302, 316, 324.
 Montana: mineral production (1911), 144.
 MOORE, CHARLES J.: *The London Mine, Mosquito Mining-District, Park County, Colo.*, xvii, 239-250.
 MOSES, ALFRED J.: *Discussion on The Sulphide Ores of Copper*, 84-85.

- MUNROE, HENRY S.: *Discussion on A Problem in Mining, together with Some Data on Tunnel-Driving*, 184.
- MURDOCH, JOSEPH, and GRATON, L. C.: *The Sulphide Ores of Copper. Some Results of Microscopic Study*, xvii, 26-81, 88-92.
- Natural gas. production of U. S. by States (1911), 144.
- Natural vs. producer gas for open-hearth furnaces, 352.
- Nevada: mineral production (1911), 144.
- New Design of Open-Hearth Steel-Furnace Using Producer-Gas* (MILLER), xix, 346-351; *Discussion* (HIBBARD), 351; (MILLER), 352-354.
- New Jersey: mineral production (1911), 144.
- New Mexico: mineral production (1911), 144.
- New York: mineral production (1911), 144.
- New York Central Lines: specifications for open-hearth steel rails, 442.
- New York meeting, February, 1913, xv.
- Nicolite:
 analysis, 229.
 effect in precipitating silver from sulphate and gold from chloride solutions, 227.
 reactions with silver sulphate solutions, 229.
- Nickel and cobalt arsenides: association with native silver, 235.
- Nodulized ore and flue dust: microstructure, 340.
- NORRIS, R. V.: *Discussion on Valuation of Iron-Mines*, 300-303.
- North Star gold mine, Grass Valley, Cal.: decrease of water with depth, 21.
- Notes on Cast-Iron* (SAUVEUR), xix, 355-376: *Discussion* (BIRKINBINE), 376; (GRIF-FIN), 383-386; (HOWE), 376-380; (JOHNSON), 386-392; (MOLDENKE), 380-383; (SAUVEUR), 392-399.
- Notes on the Formation of Ferrites in Roasting Blende* (BROOKS), xvii, 210-223.
- Nova Scotia: fire-clay deposits, 125.
- Officers of the Institute, vii.
- Ohio: mineral production (1911), 144.
- Oklahoma: mineral production (1911), 144.
- Open-hearth steel: see Steel.
- Ore deposits: gold and silver: effect of sulphides in secondary enrichment, 30, 224-238.
- Ore-treatment:
 grinding: Hardinge pebble mill vs. Chilean, 194-209.
 roasting: formation of ferrites, 210-223.
 sampling: gold-ore, 251-255.
- Ores (see also names of metals):
 microscopic study, 26-93.
 tonnage mined in U. S. (1911), 137.
- Osmond, Floris: biographical notice, 272-281.
 bibliography of scientific writings, 278.
- Oxides: iron and zinc: formation of ferrites in roasting, [214].
- PALMER, CHASE, and BASTIN, EDSON S.: *The Role of Certain Metallic Minerals in Precipitating Silver and Gold*, xvi, 224-238.
- Pania mica-mine, Hazaribagh district, India, [98].
- PARKER, EDWARD W.: *The Geographical Distribution of Mining Development in the United States*, xvi, 137-146.
 Discussion on Valuation of Iron-Mines, 303.
- PAYNE, C. Q.: *Discussion on The Sulphide Ores of Copper*, 92-93.
- Pebble mills: Hardinge vs. Chilean, 194-209.
- Pennsylvania: mineral production (1911), 144.
- Petersen and Wüst: influence of silicon on carbon content of cast-iron, [367].
- PETINOT, N.: *Discussion on Sound Steel Ingots*. 497-501.

- Petroleum: production of U. S. by States (1911), 144.
- Phosphorus: segregation in steel ingots, 411, 439.
- Phutlai Bogda mica-mine, Hazaribagh district, India, [98].
- Piping and Segregation of Ingots of Steel and Ductility-Tests for Open-Hearth Steel Rails* (DUDLEY), xviii, 424-452.
- Piping in steel: see Steel ingots.
- Plant for Hadfield Method of Producing Sound Steel Ingots* (HADFIELD), xviii, 473-478.
- Precipitating gold and silver from solutions: rôle of metallic minerals, 224-238.
- Problem in Mining, together with Some Data on Tunnel-Driving* (SIMONDS and BURNS), xvi, 147-180: *Discussion* (KEMP), 184; (MUNROE), 184; (SAUNDERS), 180-184; (SIMONDS and BURNS), 184-188.
- Proceedings of the New York meeting, February, 1913, xv.
- Production of Solid Steel Ingots* (TALBOT), xviii, 400-405; *Discussion* (CUSHING), 409-410; (DUDLEY), 408-409; (FOWLER), 411; (HIBBARD), 406, 418-419; (HOWE), 406, 413-418; (KENNEY), 407-408; (KIRCHHOFF), 413; (STEVENSON), 407, 412-413; (STOUGHTON), 412; (TALBOT), 406-408, 412-414, 419-423; (WALKER), 412; (WEBSTER), 406; (WICKHORST), 410-411.
- Puddle cinder: microstructure, 332.
- Puppe, Dr.: production of sound steel, 492.
- Pyrite:
- effect in precipitating gold from chloride solution, 232.
 - microstructure, 36.
- Pyrites cinder: microstructure, 339, 341.
- Pyritic deposits, Huelva, Spain, [236].
- Pyrrhotite:
- effect in precipitating silver from sulphate and gold from chloride solutions, 227.
 - microstructure, 41.
- Rail manufacture: mill practice, 430, 523-527.
- Rails:
- fracture a criterion of quality, 514.
 - from compressed steel ingot: analyses, 404.
 - open-hearth steel: ductility tests, 441, 448-452.
 - reports of tests, 448-452.
 - specifications for composition and testing, 442
 - split and piped: effect of size and length of ingot in preventing, 428.
 - effect of position in ingot, 431.
 - Talbot process steel: physical tests, 422.
- Rain water:
- disposal of, 5.
 - run-off of Mississippi drainage basin, 5.
- Ransome, F. L.: criteria of downward sulphide enrichment, [53]
- Rawley Mining Co.: form of daily tunneling report, 157.
- Rawley silver-copper mine, Saguache county, Colo., 147-188.
- Refractories: Canadian fire-clay deposits, 123-136.
- Revenue (California) gold-mine, Elk City district, Idaho, 119.
- Ries, Heinrich: *Fire-Clay Deposits of Canada*, xvii, 123-136.
- Rocks:
- absorption by magmas, 12.
 - contact phenomena, 14.
 - flow under compression, 19.
 - gases in, 15.
 - source of contained water, 8
 - strength under stress, 19.
- Rôle of Certain Metallic Minerals in Precipitating Silver and Gold* (PALMER and BASTIN), xvi, 224-238.

- SAGUE, J. E.: *Discussion on Sound Steel Ingots*, 487-490.
- Sales, Reno: association of native gold with sulphide copper ore, [236].
secondary sulphide enrichment, [30].
- Sampling gold-ore, 251-255.
- Sand: production of U. S. by States (1911), 145.
- Sankawa mica-mine, Nellore district, India, 110.
- Saskatchewan, Canada: fire-clay deposits, 131.
- SAUNDERS, W. L.: *Discussion on A Problem in Mining, together with Some Data on Tunnel-Driving*, 180-184.
- SAUVEUR, ALBERT: *Biographical Notice of Floris Osmond*, xix, 272-281.
Notes on Cast-Iron, xix, 355-376.
Discussion on Sound Steel Ingots, 481-482
- Savage mine, Comstock lode: ground-water level, 7.
- Schneider: action of metallic sulphides in silver nitrate solutions, 230.
- School Laboratory-Work: Sampling of an Ore Containing Coarse Gold* (LOCKE), xvii, 251-255.
- Secondary enrichment of ore deposits: effect of sulphides in precipitating gold and silver, 224-238.
- See, J. T.: theory of earthquakes and mountain formation, 12.
- Segregation:
in steel, 424-441.
negative, or impoverishment, 413, 418, 497.
- Shore: effect of temperature of quenching on hardness of steel, 522.
- Silicon:
effect on carbon content of cast-iron, 362.
segregation in steel ingots, 440.
- Silicon, aluminum, and ferro-titanium as deoxidizers of steel, 401, 428, 436, 497.
- Silver:
association of native . with cobalt and nickel arsenides, 235.
with copper sulphide ores, 232.
determination by Gay-Lussac method, 256-271.
precipitating by metallic minerals, 224-238.
production of U. S. by States (1911), 145.
- Silver-copper mines: Rawley, Saguache county, Colo., 147-188.
- Silver salts: relative solubility in water, 227.
- Silver sulphate: reactions with chalcocite, 231.
- SIMONDS, F. M., and BURNS, E. Z.: *A Problem in Mining, together with Some Data on Tunnel-Driving*, xvi, 147-180, 184-188.
- Sintered iron-bearing materials: microstructure, 330-345.
- Skey, W.: electro-motive power of metallic sulphides, [225].
precipitation of gold and silver by metallic minerals, [225].
- Slag: analyses, 329.
- Smaltite: effect in precipitating silver from sulphate and gold from chloride solutions, 227.
- SOUDER, HARRISON: *Discussion on Valuation of Iron-Mines*, 298.
- South Dakota: mineral production (1911), 144.
- South Fork gold-mine, Elk City district, Idaho, [120].
- Spencer, A. C.: secondary enrichment of ores, [65].
- STEAD, JOHN E.: *Discussion on Sound Steel Ingots*, 509-513.
- Steel: behavior of cementite in, 375.
Bessemer: formation of blow-holes, 424.
boiler-plate: blow-holes, [424].
effect of temperature of quenching on hardness, 522.
fire-box: blow-holes, [424].
"green": rolling, 406.

Steel.—Continued.

- influence of heating on crystallization, 520.
- lag in transformation : heating to high temperature increases, 516-522.
- negative segregation, or impoverishment, 413, 418, 497.
- open-hearth : analyses, 438-441
 - segregation, 436.
- thermit treated, 460.

Steel-furnaces : open-hearth, for producer-gas, 346-354.

Steel ingots :

- accelerated cooling : effect of, 461-472
- blow-holes : eliminated by deoxidizers, 402, 408, 410,
 - not eliminated by compression, 402.
- blow-holes vs. piping, 502.
- casting : large vs. small end up, 427.
 - large vs. small heats, 401.
 - large vs. small ingots, 400, 428.
 - top vs. bottom pouring, 401.
- compression of liquid . effect on segregation, 402, 415, 433.
- crucible : producing sound, by compression, 486.
- crystallization : growth in ingots, 412, 415, 509.
- deoxidizers : aluminum, silicon, ferro-titanium, 401, 436, 497.
 - thermit, 453-460.
- "green" steel : rolling, 406.
- piping : cause and cure, 418.
 - formation, 425.
 - prevention, 427.
 - by accelerated cooling of ingots, 461-472.
 - by retarded cooling of ingots, 473-478.
 - by use of thermit, 453-460.
- segregation, 436.
 - effect of size and length, 429.
 - carbon distribution, 438.
 - manganese distribution, 438.
 - phosphorus distribution, 411, 439.
 - silicon distribution, 440.
 - sulphur distribution, 439.
- separation of hard and soft layers, 412, 433.
- shrinkage : exterior and interior, 426.

Steel rails : see Rails.

STEVENSON, A. A. : *Discussion on The Production of Solid Steel Ingots*, 406, 412-413

Stibnite : effect in precipitating gold from chloride solution, 232.

Stone : production of U. S. by States (1911). 144.

"Stopping" by magmas, 13.

STOUGHTON, BRADLEY : *Discussions : on The Production of Solid Steel Ingots*, 412.
on Sound Steel Ingots, 506.

Sulphide enrichment of ore deposits, [30], [53], 224-238.

Sulphide ores : association with native silver and gold, 232-236.

Sulphide Ores of Copper. Some Results of Microscopic Study (GRATON and MURDOCH),
 xvii, 26-81 ; *Discussion* (BASTIN), 86-87 ; (GRATON), 88-92 ; (KEMP), 82-84 ;
 (MOSES), 84-85 ; (PAYNE), 92-93 ; (WINCHELL), 91.

Sulphides : oxidation by electric currents, 225.

Sulphur : segregation in steel ingots, 439.

Sulphur prints, 403.

- method of making, 414.

TALBOT, BENJAMIN : *The Production of Solid Steel Ingots*, xviii, 400-423.

Taxation :

of coal-lands, 300

influence on iron-ore costs, 299, 313, 324.

Tennantite :

effect in precipitating silver from sulphate and gold from chloride solutions, 227.

microstructure, 61.

Tennessee : mineral production (1911), 144.

Tetrahedrite : microstructure, 61.

Texas : mineral production (1911), 144.

Thermit, anti-piping : use in casting steel ingots, 453-460.

Tigre Mining Co. of Mexico, El Tigre, Sonora : electric power installation, 189-193.

Torna mica-mine, Hazaribagh district, India, [98].

Tube mills : Hardinge, 194-209

Tunnels, drainage :

Arizona Copper Co., Morenci, Ariz. : progress, [182].

Hawley mine. Sagauche county, Colo. : costs, 169.

deep vs. shallow holes, 167, 181, 188.

diagram of operations, 158.

form of daily report, 157.

labor organization, 154.

sequence of operations, 155.

ventilation, 185.

Up-to-Date mine, Caribou, Colo. : association of native silver and copper sulphide ores, 234.

Use of Anti-Piping Thermit in Casting Steel Ingots (BECK), xviii, 453-460.

Utah : mineral production (1911), 144.

Vadose region, 6.

VALLAT, B. W. : *Discussion on Valuation of Iron-Mines*, 300.

Valuation of Iron-Mines (FINLAY), xix, 282-297; *Discussion* (J. L. W. BIRKINBINE), 299; (JOHN BIRKINBINE), 297-300, 303; (BRINSMADE), 322-326; (FINLAY), 321-322; (NORRIS), 300-303; (PARKER), 303; (SOUDER), 298; (VALLAT), 300; (WHITE), 304-321.

Vermont : mineral production (1911), 144.

Virginia. mineral production (1911), 144

Vision : limitations of, 26.

Vulcan iron-mine, Michigan ; analysis of mine water, 9.

WALKER, WILLIAM R. : *Discussion on The Production of Solid Steel Ingots*, 412.

Washington (State) : mineral production (1911), 144.

Water :

connate, 8

magmatic, condensed from volcanic gases, 18.

in the earth's crust, 23.

in rocks : source, 8.

Vulcan iron mine, Michigan : analysis, 9

Water level on the Comstock lode, 7.

WEBSTER, WILLIAM R. : *Discussion on The Production of Solid Steel Ingots*, 406.

Wells, R. C. : electrical potentials of minerals, [225].

West Virginia : mineral production (1911), 144.

WHITE, E. E. : *Discussion on Valuation of Iron-Mines*, 304-321.

Why Does Lag Increase with the Temperature from which Cooling Starts? (HOWE), xix, 516-522.

- WICKHORST, M. H. *Discussion on The Production of Solid Steel Ingots*, 410-411, 484-486.
- WINCHELL, HORACE V.: *Discussion on The Sulphide Ores of Copper*, 91.
secondary sulphide enrichment, [30], [65].
- Wisconsin: mineral production (1911), 144.
- Wust and Petersen: influence of silicon on carbon content of cast-iron, [367].
- Wyoming: mineral production (1911), 144.
- Yellow Jacket mine: Comstock lode: ground-water level, 7.
- ZIMMERSCHIED, K. W.: *Discussion on Sound Steel Ingots*, 504-506
- Zinc: production of U. S. by States (1911), 145.
- Zinc ferrite: physical and chemical properties, 210.

3484